Opencast Mining / Quarrying and Alluvial Mining

Proceedings of a symposium held in London on 16th to 19th November 1964

The Institution of Mining and Metallurgy/London

[illegible]
ONE CALIFORNIA
SAN FRANCISCO, CALIF. 94111

Opencast Mining, Quarrying and Alluvial Mining – Proceedings of a symposium held on 16th to 19th November 1964 at Caxton Hall, Westminster, London S.W.1, and organized by The Institution of Mining and Metallurgy (in co-operation with The Institution of Mining Engineers and The Institute of Quarrying).

The purpose of the symposium, as defined, was to describe and discuss the best current practice in opencast mining, quarrying and alluvial mining operations. Steps were taken to secure a series of contributions covering all stages of these undertakings, ranging from exploration and evaluation through the various processes of dredging, stripping, breaking ground, loading and transport to after-treatment and land restoration. Papers dealing with automation and instrumentation were especially welcomed. In addition, descriptions of certain actual operations which seemed of particular interest were sought.

Thirty-four papers were discussed during nine technical sessions under the following headings: Sand and Gravel; Alluvials; Quarrying Practice; Transport and Equipment; Miscellaneous; Planning and Control; Stripping and Restoration; and Operations.

This volume consists of the papers presented at the symposium, the contributions to discussion and the authors' replies to questions, comments and criticisms. Finally, Personal and Subject Indexes are given.

AMERICAN
ELSEVIER

AMERICAN ELSEVIER PUBLISHING
COMPANY, INC.
52 Vanderbilt Ave., New York, N.Y. 10017

PLACER AMEX INC.
ONE CALIFORNIA BUILDING
SAN FRANCISCO, CALIF. 94111

OPENCAST MINING, QUARRYING AND ALLUVIAL MINING

OPENCAST MINING, QUARRYING AND ALLUVIAL MINING

Proceedings of a Symposium organized by the Institution of Mining and Metallurgy, in co-operation with the Institution of Mining Engineers and the Institute of Quarrying, held from 16th to 19th November 1964, at Caxton Hall, Westminster, London, S.W.1

This book is published and distributed
in the United States by
AMERICAN ELSEVIER PUBLISHING COMPANY, INC.
52 Vanderbilt Avenue, New York, N.Y. 10017

THE INSTITUTION OF MINING AND METALLURGY

Founded in 1892 and incorporated by Royal Charter in 1915, for the advancement of the science and practice of Mining in respect of minerals other than coal, and of Metallurgy in respect of metals other than iron; and to afford a means for facilitating the acquisition and preservation of that knowledge which pertains to the profession of a Mining Engineer and Metallurgist.

Details of the Institution's publications and conditions for admission to membership can be obtained from the Secretary of the Institution, 44 Portland Place, London, W.1.

The Institution as a body is not responsible for the statements made or opinions expressed in any of its publications

Copyright is reserved

Made in Great Britain at the Pitman Press, Bath

CONTENTS

page
Organizing Committee x
Foreword xi
Official Opening by The Rt. Hon. Lord Fleck, K.B.E., F.R.S. . xiii

GROUP I—ALLUVIAL DEPOSITS (INCLUDING SAND AND GRAVEL)

SESSION 1—SAND AND GRAVEL

Paper No.
1. Location and Evaluation of Sand and Gravel Deposits by Geophysical Methods and Drilling
G. VANN 3
2. Use of Modern Floating Grab Dredges for Sand and Gravel Extraction in the Upper Rhine Valley
F. KONZ 20
3. Some Developments in Connection with the Use of Suction Dredges for Mining Alluvial Deposits
J. C. C. HILL 39
4. Gravel Operations in the Trent Valley, Great Britain
R. WHITESIDE, T. D. PATERSON AND I. H. MCLEAN . 57
Discussion (Papers 1–4) 74
Authors' Replies (Papers 1–4) 86

SESSION 2—ALLUVIALS

5. Valuation of Alluvial Tin Deposits in Malaya (with special reference to exploitation by dredging)
J. K. BROADHURST AND D. J. BATZER 97
6. General Practices on Large-scale Hydraulic Mines in Malaysia
S. PAKIANATHAN AND P. SIMPSON 114

7. Mining Offshore Alluvials
M. J. Cruickshank 125

8. Gold and Platinum Dredging in Colombia and Bolivia, South America
Patrick H. O'Neill 156

9. Operations of the Yukon Consolidated Gold Corporation, Canada
W. H. S. McFarland 180

Discussion (Papers 5–9) 196

Authors' Replies (Papers 5–9) 216

GROUP II—QUARRYING AND OPENCAST MINING OF MASSIVE DEPOSITS

Session 3—Quarrying Practice

10. Developments in Blasting Techniques in Opencast Mining and Quarrying
B. J. Kochanowsky 227

11. Calculation and Application of Heading and Deep-hole Blasts for the Development of New Quarry Faces
J. Lukeš 240

12. The Limestone Quarrying Operations of Imperial Chemical Industries, Limited, Great Britain
R. G. Jackson 260

Discussion (Papers 10–12) 279

Authors' Replies (Papers 10–12) 290

Session 4—Transport and Equipment

13. Loading and Haulage in Quarries
P. Flachsenberg 299

14. Loading and Transport: Experience at Mount Morgan, Queensland
G. Sheil 324

15. Opencast Mining of Taconites in Krivoy Rog, Ukraine
M. G. Novozhilov 344
Discussion (Papers 13–15) 351
Authors' Replies (Papers 13–15) 359

GROUP III—MISCELLANEOUS

Session 5—Computation, Soil Mechanics, Groundwater and Work Study

16. Use of a Computer for Grading Iron Ore at Sept-Iles, Quebec
R. W. Kirkland and R. H. Waring . . . 363
17. Applications of Rock and Soil Mechanics to Surface Mining
A. C. Meigh and D. J. Henkel 369
18. Groundwater Control in Opencast Mining
S. C. Brealey 390
19. Two Examples of Work Study in Opencast Ironstone Mining
A. E. Ball and J. H. Daniells 416
Discussion (Papers 16–19) 429
Authors' Replies (Papers 16–19) 441

GROUP IV—OPENCAST MINING
(BEDDED DEPOSITS AND GENERAL QUESTIONS)

Session 6—Planning and Control

20. Development of the Kedia d'Idjil Orebodies (S.A. des Mines de Fer de Mauritanie) MIFERMA Islamic Republic of Mauritania
J. Audibert, P. Caruel and A. Choubersky . . 449
21. Determination of Volumes in Opencast Mining by Aerial Surveys
R. F. Rawiel 483
22. Volume Assessment by Photogrammetric Methods
H. G. Dawe 493

23. Electrical Planning for Large Opencast Mines
T. ATKINSON 505

Discussion (Papers 20–23) 532

Authors' Replies (Papers 20–23) 539

SESSION 7—STRIPPING AND RESTORATION

24. Stripping of Rock Overburden
T. M. DOVER 551

25. Economic Comparison and Detailed Discussion of Dragline versus Shovel for Removal of Overburden
R. W. ZEINDLER 565

26. Use of Walking Draglines for Intensive Opencast Coal and Ore Mining
N. V. MEL'NIKOV AND N. N. MEL'NIKOV . . . 575

27. Restoration of Ironstone Workings in the Midlands of Great Britain
T. W. JONES 584

Discussion (Papers 24–27) 601

Authors' Replies (Papers 24–27) 610

SESSION 8—OPERATIONS

28. Opencast Coal Mining in Great Britain: Operation of Acorn Bank and Westfield Sites
W. G. MARTIN 617

29. Opencast Technique in German Brown Coal Mining
W. TILMANN 637

30. Lighting of Opencast Coal Mines (with an appendix on the lighting of railway sidings)
W. B. BELL AND P. E. O'SULLIVAN 658

Discussion (Papers 28–30) 667

Authors' Replies (Papers 28–30) 676

SESSION 9—OPERATIONS

31. Selective Mining Practice at Mary Kathleen Uranium Mine, Queensland
D. L. MUNRO 681

32. Stripping Operations of the Demerara Bauxite Co., Ltd., British Guiana
W. FORBES 693

33. Ranstad—A New Swedish Opencast Mine
G. OLSSON 711

34. Open-pit Mining at Chuquicamata
R. C. BECKER AND E. W. WITCOMB 727

Discussion (Papers 31–34) 742

Authors' Replies (Papers 31–34) 752

Remarks at Closing Session 755

Personal Index 759

Subject Index 763

ORGANIZING COMMITTEE FOR THE SYMPOSIUM ON OPENCAST MINING, QUARRYING AND ALLUVIAL MINING

Professor J. H. Taylor, *Chairman*

Mr. J. T. Chappel

Mr. J. E. Denyer

Mr. T. M. Dover

Mr. E. F. Elkan

Mr. B. G. Fish (representing the Institute of Quarrying)

Mr. D. Gill (deceased)

Mr. R. Glossop

Mr. D. G. Hemmant (representing the Institution of Mining Engineers)

Mr. R. N. Pryor

Mr. J. B. Simpson

Mr. A. R. O. Williams

FOREWORD

The decision of the Council of the Institution of Mining and Metallurgy to hold a symposium on Opencast Mining, Quarrying and Alluvial Mining was taken in March 1962. An Organizing Committee was formed and the Institution of Mining Engineers and the Institute of Quarrying were invited to nominate representatives to serve upon it.

The purpose of the symposium, as defined, was to describe and discuss the best current practice in opencast mining, quarrying and alluvial mining operations. Steps were taken to secure a series of contributions covering all stages of these undertakings, ranging from exploration and evaluation through the various processes of dredging, stripping, breaking ground, loading and transport to after-treatment and land restoration. Papers dealing with automation and instrumentation were especially welcomed. In addition, descriptions of certain actual operations which seemed of particular interest were sought.

The 34 papers included in this volume represent the final selection from those submitted. In order to keep the volume to a reasonable size and price it was not possible to accept all papers received or to publish all of them in the length originally submitted. The Committee's sincere thanks are due to authors who collaborated so willingly in the unpalatable task of shortening their contributions and in some cases of reducing the number of illustrations. They are also much indebted to their many advisers on the most suitable sources for contributions upon specialist topics.

On Monday, 16th November 1964, the eve of the symposium, the Sixth Sir Julius Wernher Memorial Lecture, 'Vertical integration in mining industry', was delivered by Mr. R. P. Koenig, O.B.E., President, Cerro Corporation, in Caxton Hall, Westminster. The following morning the symposium proper was officially opened in the same building by the Rt. Hon the Lord Fleck, K.B.E., F.R.S. It was attended by some 400 participants and comprised nine technical sessions, each involving discussion of three to five papers introduced *en bloc* by a rapporteur. The closing session took place on the morning of Thursday, 19th November.

The symposium was followed by short visits to opencast operations in England. These included the National Coal Board's Radar North Opencast, near Morpeth, Northumberland; the United Steel Companies' bedded ironstone workings near Colsterworth, Lincolnshire; Imperial Chemical Industries' Tunstead limestone quarry, near Buxton, Derbyshire; the granite quarry of E.C.C. Quarries, Ltd., at Croft, Leicestershire; the gravel workings of Hoveringham Gravels, Ltd., near Rainham, Essex; the Borough Green quarry in Kent of Amalgamated Roadstone Corporation; the Shoreham works in Sussex of

Associated Portland Cement; the Redhill pits in Surrey of the Fullers' Earth Union; the operations of South Coast Sand and Ballast Co., Ltd., Southampton; the Smallford site of St. Albans Sand and Gravel Co., Ltd.; and the Chertsey pit of Hall and Ham River, Ltd. The thanks of the Committee are due to all the operators who made these visits possible.

In the course of the symposium attention was drawn to the increasing percentage of world mineral output being produced by open-pit methods. The Committee hope that readers will find that this volume describes a representative sample of opencast operations and that appropriate emphasis has throughout been placed on recent technical developments.

J. H. TAYLOR
Chairman of the Organizing Committee

OFFICIAL OPENING BY LORD FLECK, K.B.E., F.R.S.

The Symposium on Opencast Mining, Quarrying and Alluvial Mining was held at Caxton Hall, London, from 16th to 19th November 1964. The Symposium was organized by the Institution of Mining and Metallurgy, in co-operation with the Institution of Mining Engineers and the Institute of Quarrying. The President of the Institution of Mining and Metallurgy for the Session 1964–65, Mr. D. S. Burwood, was in the Chair.

Mr. D. S. Burwood, opening the proceedings, said: This symposium, as you know, has been organized by the Institution of Mining and Metallurgy in association with the Institution of Mining Engineers and the Institute of Quarrying, whose Presidents, Mr. G. C. Payne, and Mr. F. Frith will be taking the Chair at subsequent sessions. We are fortunate in that Lord Fleck has accepted the invitation to open our symposium. Lord Fleck, as is known the world over, is a distinguished administrator and Chairman of Imperial Chemical Industries. He is also a distinguished scientist in his own right and a Vice-President of the Royal Society. He is an Honorary Member of the Royal Society of Edinburgh and has many other academic distinctions; we in the Institution of Mining and Metallurgy are glad to number him among our own Honorary Members.

The Rt. Hon. the Lord Fleck, K.B.E., F.R.S.: I feel deeply honoured that you, Mr. President, your Past-President, Professor Dunham, and the Council of your Institution should do me the honour to ask me to open your Symposium on Opencast Mining, Quarrying and Alluvial Mining. I will say right away that I have no practical experience in, or specific knowledge of, opencast mining, but that has not prevented me from taking a considerable interest in this department of technology for quite a number of years.

I can assure you that I have seen sufficient operations to be duly impressed with the amount of logistics, the amount of care and the amount of solid radical work that go into opencast mining. Rather than diminishing, opencast mining has become more and more industrially and economically important.

I would just mention the three operations which have brought home to me the importance of opencast mining. First, I was impressed by the work in Kiruna, North Sweden, and by almost the only economically viable railway in the world. Secondly, I would also take this opportunity to pay tribute to what is being done in India in the opencast workings at Neyveli, 150 miles south of Madras. There is something, I am told, of the order of several million tons of brown coal at

Neyveli 150 ft down, and the organizational abilities being displayed there are most striking. This morning I have learnt that one of your papers deals with the water problems that they have had—and presumably are still having. I am sure that the Neyveli opencast operations will have a profound influence on the whole economy of India. It is one of the most important projects being undertaken at the present time.

Thirdly, I cannot refrain from expressing admiration for what our own National Coal Board are doing. Some three months ago I went over the Maesgwyn opencast near Swansea, South Wales. This work, too, is of considerable value and, again, requires first-rate planning.

These three operations show the need for organization in mining and this is one of the reasons why I was very glad that the Institution of Mining and Metallurgy has arranged the present symposium. Many will appreciate its value.

I have not spoken about quarrying; frankly, I know less about quarrying than I do about opencast mining, apart from appreciating that quarrying is a very important matter for the whole of Derbyshire and that it requires careful logistic consideration.

Opencast working also has its importance from the farming point of view. Those of us who have seen the restoration carried out by the National Coal Board in this country will, of course, realize this. I venture to think, again not being an expert but merely, I hope, a reasonably accurate observer, that the replaced soil is actually better from an agricultural point of view than the original. I base this remark on what I have seen in the County of Durham.

It is with great pleasure that I declare this symposium open. I believe that your discussions will be of great value to you and to the mining community as a whole and I wish the symposium every success.

Mr. D. S. Burwood, in reply, said: I am sure it is your wish, on my behalf and yours, that I should thank Lord Fleck for the interesting and gracious way in which he has opened our symposium. Your remarks, Sir, concerning the subject of the symposium will, I am sure, do much to stimulate a great deal of valuable discussion.

GROUP I—ALLUVIAL DEPOSITS
(including sand and gravel)

Session 1—Sand and Gravel (Papers 1–4)

Chairman: Mr. D. S. Burwood *Rapporteur:* Mr. P. R. Harris

550.8:553.623/.624

Paper 1

Location and Evaluation of Sand and Gravel Deposits by Geophysical Methods and Drilling

G. VANN, B.Sc., A.M.I.Q., F.G.S.
Director, Le Grand, Sutcliff and Gell, Ltd., Southall, Middlesex, Great Britain

---

SYNOPSIS

The planning of reconnaissance and detail surveys for sand and gravel is discussed and an outline of the electrical resistivity and shallow seismic geophysical methods is given.

The uses of geophysical methods to plan or to supplement proving and sampling borings, with the resultant reduction in exploration costs and the greater accuracy and detail of results, are explained.

Mention is made of various types of drilling useful in gravel surveys, cost guides are given, and the presentation of data preparatory to the purchase of land or the planning of quarry development is outlined.

The location of the principal sand- and gravel-bearing areas in the United Kingdom is known, but variations within these areas are such that it is seldom possible to be certain that sand and gravel are present under any particular site.

In many underdeveloped overseas areas, particularly in areas of dense vegetation, there may be little or no surface evidence of the presence of gravels.

Because of the nature of deposition of sands and gravels, and the erosion and re-deposition that may occur in their subsequent geological history, these deposits are likely to vary in thickness and quality, thereby presenting exploration and evaluation problems not encountered in dealing with other non-metallic and industrial minerals and rocks.

Three main types of deposit can be recognized. *River deposits*—either flood plain or terrace—are generally the easiest to locate and the least variable in thickness and quality, but buried channels of thick material, washouts, etc., are difficult to find by conventional methods. *Glacial and fluvioglacial deposits*, from the nature of deposition, are likely to be highly irregular in quality and thickness and may be buried under a considerable thickness of overburden—which makes detection difficult and expensive. So far as *bedded deposits* are concerned, pebble beds occur in sandstones, for example the Bunter deposits of the Trias in the United Kingdom, but are likely to occur in irregularly shaped lenses.

Each type of deposit has its own special exploration problems, but in terms of drilling and geophysics there are many common points.

RECONNAISSANCE EXPLORATION

Although it may not be recognized as a separate phase, the initial work on a new site should be considered as a reconnaissance examination. Where the site to be surveyed does not belong to the organization having the work carried out—a normal state of affairs in the case of a sand and gravel company—any one or all of the following points may apply:

(*a*) time to carry out the investigation may be limited by reason of expiry of option to purchase, auction date, etc.;
(*b*) access to parts of the site may be restricted by reason of valuable crops, tenant objections, marshy or flooded ground, etc.; and
(*c*) the cost of executing the survey may be restricted for normal commercial reasons or by agreement with the owner of the land, who may be sharing the cost.

The application of geophysical survey techniques is relevant in all these considerations because the methods allow large areas of ground to be covered quite quickly and economically. Assuming a deposit with base not more than 20 to 30 ft below surface, electrical resistivity observations can cover on average up to 60 acres per day in reconnaissance fashion, that is approximately one observation per four acres.

The work is carried out using light but robust and easily operated equipment not requiring the use of a vehicle on site (see Figs. 1 and 2).

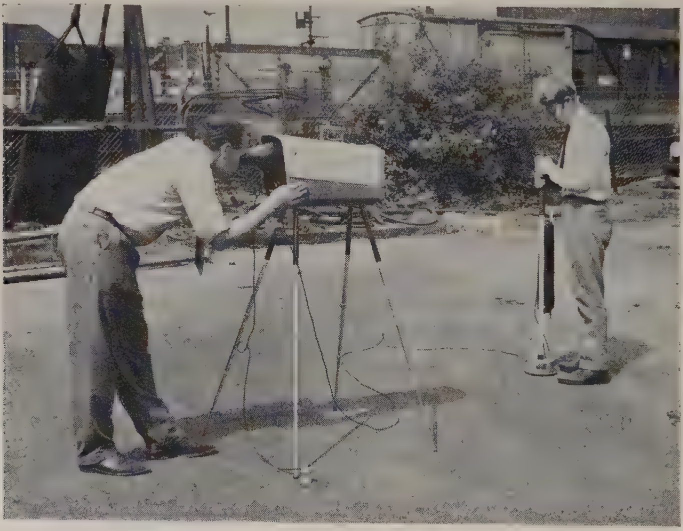


Fig. 1.—A portable shallow seismic refraction unit in operation; the seismic shock wave is created by the tamper on the right, picked up by the geophone in the foreground and displayed on the cathode-ray screen of the instrument on the left.

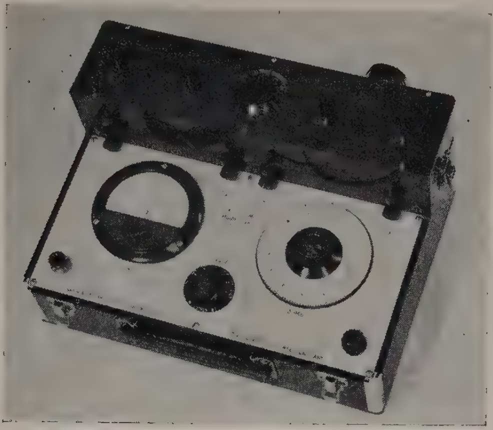
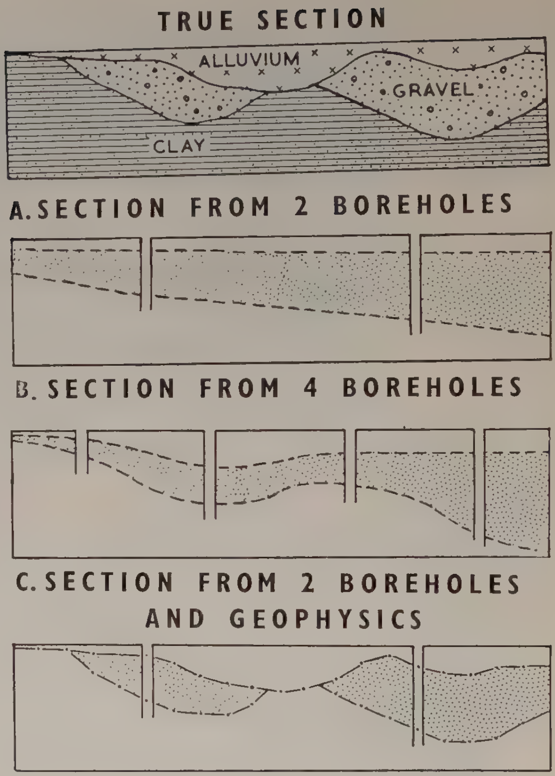


Fig. 2.—An electrical resistivity unit powered by internal dry batteries.

Normally no damage is caused to the ground surface or to crops—or at most only that brought about by one or two people walking across the site. Considerably more information can be gained for the same cost compared with that obtained using conventional techniques (Fig. 3).

It must be borne in mind, however, that at the reconnaissance stage geophysics is largely a tool giving a yes/no answer to the question 'Is sand and gravel present?' It may also give an approximate idea of overburden and productive layer thicknesses, but it will not normally give any idea of quality or distinguish between sand, ballast and hoggin. Quality information can only be obtained by testing samples from trial borings or pits—such correlation being necessary in any case to gain maximum value from the geophysics.

The use of geophysics as a reconnaissance tool provides sufficient information for a decision on whether expenditure on drilling is justified, and also will enable bore-hole locations to be selected. Geophysics also has a very important role in looking for potential sites in a given area. Obtaining options on sites may be a lengthy and somewhat expensive business, particularly if the owner insists on complicated methods of assessing the value of the mineral content, and much effort and money may be wasted. Exploration exercises leading to negative results safeguard the interests of the prospective purchaser, but quarrying concerns live by producing material—not by proving where it does not occur. In many cases a few judicious geophysical tests may show quite quickly and cheaply if an area is worthy of further attention. The owner of the land may be amenable to such tests being carried out (no damage is caused) without the formality of an option, whereas he would object to bore-holes being drilled.



TRUE SECTION

A. SECTION FROM 2 BOREHOLES

Boring evidence suggests that gravel and overburden thicknesses are reasonably uniform along the section. Other bore-hole locations could give equally misleading results

B. SECTION FROM 4 BOREHOLES

A slightly more accurate section has been obtained at considerably greater cost

C. SECTION FROM 2 BOREHOLES AND GEOPHYSICS

A sufficiently accurate picture for commercial purposes has been obtained at a reasonable cost

Method	*Time taken, days*	*Approximate cost, £*	*Volume overburden, yd^3*	*Per cent error*	*Volume gravel yd^3*	*Per cent error*
True section	—	—	100 000	—	159 000	—
A	4	120	70 000	−30	244 000	+50
B	8	240	91 000	−9	189 000	+20
C	4	200	94 000	−6	165 000	+10

Fig. 3.

DETAILED SURVEY

If the reconnaissance survey shows that the site warrants further investigation, in most cases it would be desirable to apply detailed geophysics and then to put down trial borings or pits. Where time is important, however, these two aspects of the work can be carried out concurrently—indeed, where time rather than cost is the primary consideration drilling and preliminary geophysical survey may begin at the same time. There is little or no objection to this where fairly strong evidence exists that productive material is likely to be present, but it may lead to time and money being wasted in putting down borings in barren areas. Although some operators require proof of negative conditions, the geophysical evidence would usually be used to make sure that all borings encounter productive material because a negative boring may cost just as much and require as much time as one giving good results.

Experience in the United Kingdom has shown that, except in the most variable situations, an average intensity of one geophysical observation per acre coupled with one bore-hole or trial pit per ten acres is a reasonable compromise between cost and accuracy, for small- and medium-size sites (say up to 500 acres). For larger sites, particularly those of several thousand acres, half of the above intensities may be acceptable, but they should be considered as absolute minima.

GEOPHYSICAL METHODS

In the fields of oil and metal mining the value of geophysical exploration has been recognized for almost half a century. The geologist, petroleum or mining engineer who has to select a position for putting down a bore-hole, which may cost thousands or indeed millions of pounds, will obviously use every known tool to reduce the risk of a negative result to the minimum, particularly when the more obvious deposits and simpler structures have been located and explored. In these industries geophysical exploration is a recognized scientific method and its techniques and instrumentation are undergoing constant improvement.

In the non-metallic mining field, however, and in particular in the sand and gravel industry, only in the last 15 to 20 years has the aid of geophysics been enlisted to any extent—almost certainly because the smaller operators have been content to use traditional methods so long as they appeared adequate, and quarry and exploration managers have but the briefest of contacts with geophysics during their training.

No visual method or shallow boring technique can show that a 50-ft thickness of sand and gravel lies buried under 30 ft of overburden. Who will be responsible for continuing a boring which has already penetrated 20 ft of clay without some guide to the fact that thick sand and gravel lies at greater depth? Only geophysics can provide this guide.

There are many well established geophysical techniques, but in the present stage of knowledge only two have any relevance to the problems of the sand and gravel industry—the electrical resistivity and shallow seismic methods. Both methods can be applied to the elucidation of small-scale problems and short-interval lateral variations and allow the depth of penetration below surface to be controlled by the way in which the observations are made.

Although the methods are well established and are treated in detail in textbooks, many popular misconceptions about them exist and certain important points are not generally known. The following brief descriptions may be useful.

Electrical Resistivity Method

The resistivity of a material is the resistance of a unit amount of the material and is a constant for each pure substance. Substances occurring naturally are not pure and the resistivities of rocks and minerals therefore vary over wide ranges. This is unimportant, however, as the method relies on contrasts in resistivities rather than on absolute values. At the same time this does mean that measurements of resistivity are rarely diagnostic of the materials involved, other than in very broad terms—hence the need for correlation by borings.

The resistivity of a soil or rock depends on various factors—the constituent material, the ratio of voids to solid, the shape and size of the voids, and the amount and quality of the water content, etc. It is possible to measure the variations in resistivity with depth below surface by measurements made only at the surface. Generally, a measured current (either a.c. or d.c.) is introduced into the ground between two metal electrodes and the resultant potentials measured between two other intermediate metal electrodes (for a.c.) or porous pot electrodes (for d.c.). In the simplest arrangement the four electrodes are disposed at equal intervals along a straight line; the apparent resistivity obtained by computation from the current and potential measurements relates to a depth zone below surface approximately equal to the distance between adjacent electrodes. By successive expansion of the electrode intervals (2, 4, 6, 8, 10, 15, 20, 25, 30, 40 and 50 would be a typical arrangement) the variations in apparent resistivity from surface down to a number of depths (2, 4, 6, 8, etc.) can therefore be obtained. The resistivity measurements are plotted against depth on a graph. It is often possible to see features at a boring position which correlate with the geological conditions, thus allowing interpretation of the graphs at other positions. Otherwise, the graphs can be subjected to mathematical analysis or compared with a standard set of graphs produced for a wide range of resistivity contrasts and thickness-of-strata ratios.

Normally the method can only be used on dry land. It cannot be used where insulating materials such as asphalt are present at surface, unless holes for insertion of the electrodes are broken through this layer. It is affected by the presence of metallic conductors at or under the surface. Reasonably level topographic conditions are also essential —often a difficult requirement. Space may also be a limiting factor since an electrode spread three times the depth of penetration is required, i.e. to reach 50 ft below surface a straight-line lateral spread of 150 ft is necessary.

The method is speedy and observations to 50 ft below surface can be made at twenty observation stations, at least, in a normal working day, given reasonable site and weather conditions. The cost would normally be between £20 and £30 per day.

Practical experience in areas of glacial superficial deposits, particularly in Essex, has shown that the interpretation of sand and gravel thicknesses under overburden thicknesses of 30 ft or more cannot be achieved with any reliability by conventional qualitative or mathematical methods.

Using the results obtained from twenty or more borings and over one hundred resistivity depth probes in an area of thick overburden (20–50 ft) underlain by comparable thicknesses of sand and gravel, a new system of treating the resistivity data and analysing the results has been evolved. So far the system has only been used to predict strata thicknesses in the same general area, but subsequent borings showed almost exact agreement with the predictions.

Modern earthmoving machinery is enabling large overburden thicknesses to be moved economically, compared with the alternative of longer transport distances from pit to demand area. The possibility of using geophysics to minimize the number of deep borings to prove such deposits is an important aid in the search for sand and gravel.

Seismic Methods

Refraction and reflection are the two distinct techniques comprising seismic methods. In sand and gravel surveys refraction techniques are the main concern since the instrumentation is simpler and is currently available in a form suitable for shallow surveys.

The method involves the application of the principles governing the refraction of light to the transmission of shock waves in the ground, for shallow work such waves being created by dropping a hammer or tamper. It is necessary to measure the time taken for the shock waves to travel from the point of initiation to a detector (geophone) which converts ground movements into electrical impulses which can be amplified and displayed on a cathode-ray screen or recorded on photographic paper (see Fig. 1).

The velocity of transmission of these shock waves is dependent on the density and elastic properties of the rock through which they are travelling. In general, the harder the material and the greater its depth of burial, the higher will be the velocity of transmission—a vital requirement for refraction work, which relies on waves that have travelled down into and along a rock layer with a higher velocity of transmission than the layer above it, travelling back to surface again and reaching the detector before the waves travelling in the upper, slower horizons.

The spacing between the hammer and detector is increased progressively (say at 5- or 10-ft intervals) until a distance is reached at which the first energy to arrive at the detector is from the depth zone of interest. The travel times are plotted against distance on a graph and, from the velocities and other information which can be obtained from the graph, depths to various interfaces and continuous profiles of these interfaces can be obtained. As with resistivities, seismic velocities are not necessarily diagnostic of the materials involved.

The seismic method can be used on land or under water. Thin layers of man-made materials, such as concrete or asphalt, usually do not interfere with observations, but the method will obviously fail if the near-surface layer has a higher velocity than any of the underlying strata. Extreme topographic variations can be troublesome, but can generally be compensated for if accurate levels are available. As with resistivity, straight-line observations are generally required and therefore a reasonable amount of open space is necessary. As a very broad generalization, the distance required will be about four times the depth of the strata interface to be investigated.

The cost of undertaking shallow seismic observations using a tamper as source of energy averages £25 to £30 per day, during which time readings at ten to fifteen stations with maximum tamper to detector distances not exceeding 100 ft can be accomplished. Man-made seismic disturbance (traffic, machinery, etc.) or natural disturbances (wind in trees, waves on a shoreline) may make observations difficult, but it is usually possible to overcome such problems by waiting for the disturbance to cease.

Choice of Geophysical Method

Seismic observations take longer than electrical observations and thus it is generally preferable to use the latter in order to achieve greater speed and economy. The use of electrical resistivity is particularly suitable where the sand and gravel layer is overlain and underlain by clay since the resistivity contrasts in this situation should be well marked. Conversely, in this situation seismic refraction is unlikely to be of use as there will almost certainly be a velocity inversion between the gravel and the clay beneath, which prevents the detection of this interface.

If the sand and gravel rests on hard rock, however, such as sandstone, rock marl, limestone or chalk, there may be very little or no resistivity contrast, but there is a strong possibility that a seismic contrast will be found.

The location of pebble bands or lenses in sandstones is difficult by either geophysical method and at present there is no reliable technique that can guarantee success. The presence of the pebbles sometimes gives rise to relatively high resistivity or seismic values, but by no means in every case. Even if a contrast does exist the work is not likely to enable thicknesses to be determined—it usually only makes recognition of the presence of the stones possible and allows the lateral distribution to be defined. Even if nothing more is achieved, it is of assistance since boring in hard formations is costly and barren holes should be avoided.

DRILLING

As the various methods of carrying out physical examination of subsurface conditions and samples are well known only the following notes are presented—as a guide to selection and probable costs. The latter are for the work undertaken in the United Kingdom on a commercial basis by an independent drilling specialist.

Percussion Methods

Percussion methods cover techniques other than rotary drilling and, as such, include the use of tools, other than bits or chisels, which are in fact used in a percussive manner. The tools most commonly used for penetrating sands and gravels are shells which loosen the material and bring it to surface, at the same time allowing the lining tubes to enter the stratum. Bits or chisels may be required to break up cobbles or boulders larger than the shell or casing.

Properly executed borings generally should be not less than 4 or 5 in. in diameter and, because of restrictions imposed by size of drilling rigs and operating costs, would not be more than 10 in. Drilling at smaller diameter is undesirable as it is then impossible to obtain samples including the very large diameter stone, and progress is slower and costs rise if much time has to be spent in breaking up such material.

Normally the tools are used on rods, which enables accurate measurements of depths and thicknesses to be obtained, but for greater speed may be used on cable. Another advantage is that the equipment can be reasonably light and portable and can be hand-operated if necessary (see Fig. 4). Power-operated equipment is small enough to be towed by a Land-Rover and therefore damage in gaining access should be negligible, except on very wet and soft ground.

In unconsolidated formations with no cohesion (such as clean sand and gravel) the bore-hole must be lined, but this does ensure that the

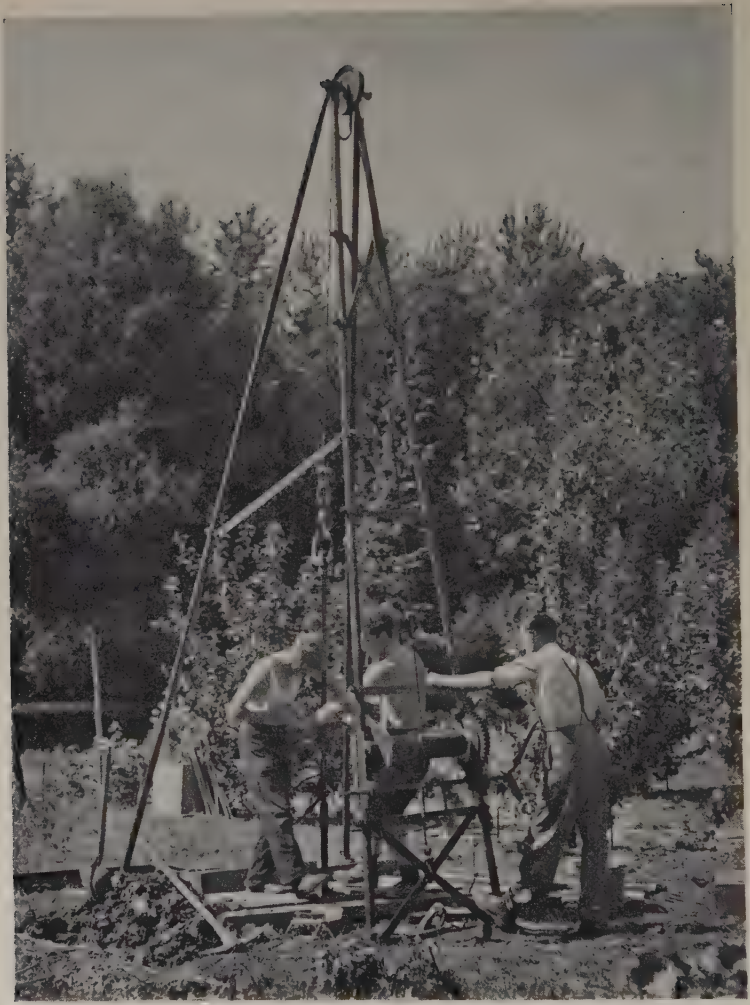


Fig. 4.—Hand-operated shell and auger drilling rig suitable for use in difficult access sites.

material being extracted is coming from the depth of the drilling tools rather than higher up the hole. On the other hand, the operation of the shell requires the presence of water and its use may cause the material brought to surface to be cleaner and to contain less fines than the *in situ* material.

Because depth of drilling is virtually unlimited, thick overburden and productive layers can be proved. Pebble layers in sandstones may also be proved by percussion drilling—in which case the chisel or bit would be necessary.

Drilling costs in superficial gravels could be expected to be between £1 and £1 10s. per foot, to which must be added positioning charges for getting to and from the site and erecting and dismantling charges at each bore-hole position. Costs might be considerably higher in consolidated formations.

At present this method is considered to be the most reliable available for proving superficial sands and gravels and is the required method for use in the United Kingdom for civil engineering foundation investigations—the only drilling use for which a code of practice has been established.

Auger Methods

Hand auger

In difficult access areas, e.g. marshes, hand-augering may be of limited use in proving thicknesses of overburden and indicating the presence of sand and gravel. In the same category, but of even less use, may be included proving with a Mackintosh tool or similar device which relies on the greater resistance of sand and gravel compared to clay (but less than solid rock) to reveal the presence of this material.

Stiff clay overburden, particularly if stony, may make penetration by hand auger or probe difficult or impossible, and generally no progress can be made by hand auger once the loose gravel or water table has been reached.

The cost of hand-augering may be in the range 8s. to 12s. per foot, to which must again be added positioning charges.

Single-flight power auger

Single-flight power augers are common accessories for agricultural tractors to enable holes for poles, fence posts, etc., to be put down. These are usually limited to depth penetrations of the order of 10 to 12 ft. Special machines, incorporating a small motor (petrol or electric), which can be carried about by one man and operated by one or two men, have been developed in the last few years and these have a slightly greater depth range (see Fig. 5).

Penetrations of up to 20 ft can be obtained when what are commonly called ‘soil-boring machines’, which are mounted on large tractors or lorries, are employed. Apart from the power which enables greater depths to be reached and more difficult materials to be penetrated, these auger machines suffer from the same disadvantages as hand-augering, i.e. that unconsolidated material, particularly under the water-table, collapses into the hole and makes all further progress impossible.

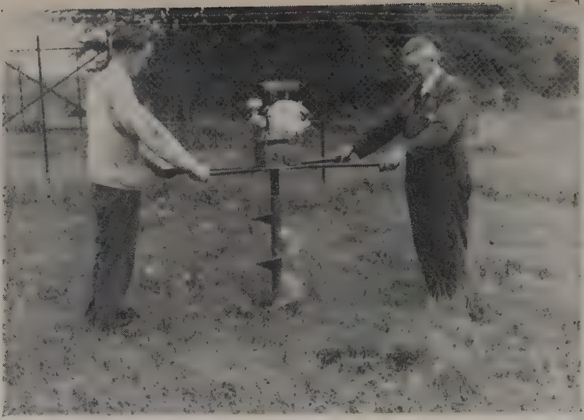


Fig. 5.—A portable power auger, driven by a small petrol engine, showing a continuous-flight auger in use.

In general, the rotary action of these machines churns the strata and tends to mix one layer with another. One particular danger is that clay overburden may be carried down into a lower sand and gravel layer, enabling the walls of the hole to stand up and at the same time creating the impression that the material is a clay with stones. The quality of the results depends to a much greater extent on the skill and experience of the operator than in other drilling techniques.

There is, however, no doubt that the use of this type of equipment provides a relatively cheap means of determining (*a*) overburden thicknesses where these do not exceed 20 ft, and (*b*) the full thickness of hoggin or dry sand and gravel deposits that will stand up without lining the bore-hole, where the base of these deposits does not exceed 20 ft or so below the surface. The cost of drilling with these machines is generally in the order of 5s. to 10s. per foot.

Continuous-flight power auger

Fitted on suitable large prime movers and driven by high-energy power units, the latest American machines can carry out continuous-flight augering to depths up to 50 ft below surface—more than adequate for the majority of gravel investigations. The great advantage is that this type can penetrate unstable strata without the need for lining tubes. In such strata it does not make holes in the normal sense of the term; the auger is rotated downwards, extra auger lengths being added at the surface until by the rate of progress the operator is aware that the full thickness of sand and gravel has been penetrated. The whole auger with its load of material is then withdrawn from the ground in stages, being rotated at the same time if necessary. Where rotation can be avoided, sufficient material can be retained on the auger spirals to provide samples.

The presence of clay under the gravels should be seen on the bottom of the auger, thus proving that the full thickness of the deposit has been penetrated. Where hard rock underlies the deposit no positive evidence of it may be afforded and difficulty in penetrating might equally arise from a large boulder in the gravel and therefore the method may sometimes give misleading results.

The disadvantages of this type of machine are that they are expensive to purchase and may cause more damage in gaining access than other types of drilling plant. Progress rates of 1 ft/min or more can, however, be achieved and if the work load is sufficiently high the initial purchase cost can be written off reasonably quickly.

Air Drilling

At present air drilling would only be of interest in proving consolidated and semi-consolidated formations where the bore-holes will stand up without lining other than a short length of conductor pipe at surface if necessary.

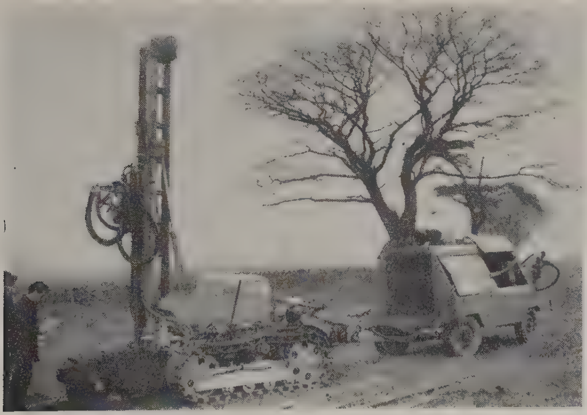


Fig. 6.—Small compressed-air-driven rig which can operate with a 'down-the-hole' hammer bit or with a rotary bit. The compressor provides air for drilling and for moving the unit on site.

Relatively small rigs (see Fig. 6) are available which can drill by percussion or rotary methods, or by a combination of the two. As the required air-compressor capacity increases rapidly with diameter of hole drilled, a hole of the order of 3-in diameter would normally be used.

One advantage is that as no water is used a sample including the full percentage of fines can be obtained at surface; however, medium- and large-diameter stone is generally broken up in the drilling process. The method can be helpful in locating pebble horizons in sandstones. If the gravel horizons are thick, air drilling may require to be supplemented by conventional or down-the-hole percussion holes once the areas of interest have been established.

Progress rates of the order of hundreds of feet per day can be achieved at costs of about 5s. per foot for the actual drilling.

Trial Pits

Except for thin deposits, trial pits are not of major use in quantitative evaluation of sand and gravel deposits. Once the water-table is encountered it is difficult to be certain that the material being brought up comes from the depth of grabbing. There is a tendency for clay to be washed away and for gravel sliding in from the sides of the holes to be brought up, thus exaggerating thicknesses.

In dry deposits of any thickness the surface dimensions of a hole necessary to enable depths of more than a few feet to be proved are such that the damage done would normally not be acceptable to the owner of the land.

The major advantages of trial pits are that they allow (*a*) a cross-section of the strata *in situ* to be inspected, and (*b*) samples of material with full percentage of fines and large stones to be obtained.

Choice of Drilling Method

In the writer's opinion, for full proving of superficial sand and gravel deposits at present the shell and auger (percussion) method provides the best combination of accuracy of recording strata changes and sampling, with reasonable cost and minimum damage in gaining access.

At the same time, however, power auger boring is useful for proving shallow deposits and as a reconnaissance tool either alone or in combination with geophysics.

DISTRIBUTION OF OBSERVATIONS

The current tendency is to locate bore-holes either on a rectangular grid at the apexes of triangles or on some other more or less fixed and regular pattern, modified as necessary by any irregularities in the shape of the site.

Where a geophysical survey has been undertaken prior to the boring, the primary consideration in selecting positions for drilling will be that of providing correlation evidence for the various types of geophysical result. Where a choice exists, however, and this is usually the case, there is a tendency to choose locations which give an even distribution across the site. The geophysical observations themselves are usually carried out at stations arranged as a grid or along a series of intersecting traverse lines. The danger in using a regular pattern is that the interval between adjacent observations may be coincident with a natural 'wavelength' of thickness variation. At first this may seem unlikely, but experience in the field has shown on several occasions that two sets of borings, undertaken on a site at different times each disposed on a similar interval but displaced grid, have given reasonably consistent results within each set, but with a significant divergence between the two sets. The use of geophysics to supplement trial borings enables a higher density of observations to be achieved and thus to overcome such difficulties.

PRESENTATION OF RESULTS

Various organizations have individual preferences in the way results should be presented. In the present writer's opinion certain minimum requirements (listed below) must be met; surprisingly, however, they are very often not found in the data available.

(*a*) A plan of the area of survey clearly indicating the position of the bore-holes, trial pits, geophysical observation stations, etc. Each observation should be clearly lettered or numbered so that at any time after completion of the work reference can be made to any particular boring, etc. The suitable scale of the plan will depend on the size of the survey area and the nature of the investigation; for small areas to be covered in detail (say up to 300 acres), 1 to 2500 is a suitable scale, while for large areas of several thousand acres or for reconnaissance surveys a scale of 6 in. to 1 mile is adequate.

(*b*) A detailed record of the strata encountered in each boring, together with the dates of boring, the size of the hole, depth of striking water and standing water-level and depths of samples. It is not sufficient to keep only a record of overburden and productive layer thicknesses; the results may be tabulated in this way for quick reference, but the full details must also be available.

(*c*) Geophysical data need not be presented in the report, but it is essential that the field books, plotted data and computation sheets should be available for inspection if required.

Diagrams

The various methods of presenting the data on diagrams, outlined next, may be used singly or in combination.

(1) Diagrammatic bore-hole sections plotted on a site plan, with or without detailed strata descriptions written beside the sections; while this method enables a lot of information to be presented on one diagram the significant trends and variations are not apparent at a glance.

(2) Written figures for overburden and productive layer thicknesses and water-levels present at each bore-hole position on the diagram; this method suffers from the same disadvantages as the first, but is easier to follow, particularly if there are large numbers of borings.

While both methods can be used for bore-hole information, when a geophysical survey has also been undertaken there is too much information for it to be recorded in these ways and one or both of the following must be adopted:

(3) cross-sections along selected lines of traverse; these may be given in their correct relative positions on the site plan or on separate diagrams. The former method is sometimes useful if a large area has been covered in a reconnaissance fashion and contour plans cannot be prepared. Where there are significant thickness variations, however, separate diagrams are a useful supplement to contour plans;

(4) isopachyte or equal thickness contours for overburden and sand and gravel thickness and contours of sand and gravel to overburden ratio.

The writer believes system (4) to be the best, even though it requires the preparation of two or three base plans of the area. Drawing office work can be minimized, however, by only showing geophysical and bore-hole positions, field boundaries and other topographical data on one plan (usually the one giving sand and gravel thicknesses) and using an outline plan for the contours of overburden and gravel to overburden ratio.

The major advantages of this method of presenting results are that (*a*) trends of the deposit are readily grasped by everyone likely to be concerned with the site, including the owner: this may be an important point for if the owner cannot understand clearly what results have been obtained he is likely to be less co-operative than otherwise; this may also be true with regard to obtaining planning permission; and (*b*) no undue significance is placed on any one result whether it be a boring or a geophysical observation.

In view of the nature of sand and gravel deposits discussed earlier, the assumption that any one boring or observation represents the conditions more than for a few feet from it may be quite false.

The process of constructing contour plans from various pieces of information allows the overall trends to appear, at the same time revealing the anomalies of any erroneous observations.

Computed Data

The report should include all the information that can be derived from the survey, including volumes of sand and gravel and overburden; average, minimum and maximum thickness of sand and gravel and overburden; average gravel and overburden yields per acre; and average sand and gravel to overburden ratio. The total survey area, barren area, if any, and productive area should be stated.

Accuracy

It is, of course, desirable to have some indication of the potential accuracy of the computed data presented. It is only possible to know how accurate the volumes given for a site were when all the material has been excavated. In the writer's own organization, which has undertaken nearly six hundred combined geophysical and drilling surveys in the twenty years since the war, such information has only been made available for a few sites—because exploration may be up to ten to twenty years ahead of commencement of exploitation and, in addition, the period of working has to be taken into account. It is to be hoped that comparative information will become increasingly available so that accuracies can be judged on factual evidence.

It appears that where boring has been undertaken at an intensity of one to ten acres, and geophysical operations at an intensity of one per acre, it is not unreasonable to anticipate a ±10 per cent accuracy, provided the geophysical and drilling results have been in good agreement and the variations in thicknesses are not too extreme.

CONCLUSIONS

The increasing use of geophysical techniques in combination with bore-holes has enabled the cost of exploration for and evaluation of sand and gravel deposits to be stabilized, at the same time giving greater accuracy.

At the time of writing, the cost of carrying out a combined geophysical and drilling survey over medium-size sites in the United Kingdom is between £5 and £10 per acre, which is the same order of cost as that ruling ten years ago.

622.234.8:622.362.3/.4(43)
621.879.36:622.362.3/.4(43)

Use of Modern Floating Grab Dredges for Sand and Gravel Extraction in the Upper Rhine Valley

F. KONZ, Dr. Ing.

Chief Engineer, Karl Epple, Kies-, Sand- und Steinwerke, Stuttgart–Bad Cannstatt;
Vice-Chairman, Bundesverband Kies-, Sand- und Moertelindustrie

---

SYNOPSIS

During the last ten years a new type of floating dredge has been developed and used with success in Germany. The present paper describes its use in the Upper Rhine Valley between Mannheim and Basle (the Oberrhein).

The Rhine Valley between the Vosges and the Black Forest is filled to a depth of about 300 ft with alluvial material. The increasing demand for sand and gravel and the scarcity of land for their production have led to the need to work the pits to greater depths with the result that plant manufacturers have co-operated in the development of the 'Schwimmtiefgreifer'—floating deep grab dredge. The grab capacity is up to 8 yd^3 and dredging can be carried out to a depth of 130 ft. Construction is simple and main wear is restricted to cables. Associated transport systems are described and some costs are given.

Before dealing with the technical problems of the use of floating dredges for sand and gravel winning in the Upper Rhine Valley it is desirable to consider the geographical and geological conditions which influence these operations. The part of the Rhine Valley in question is the stretch between Basle in the south and Mainz in the north. The area within which sand and gravel can be economically produced is shown enclosed by the two dotted lines in Fig. 1. These boundaries do not strictly follow the geological deposits, but are influenced by factors such as the agricultural value of the land, etc.

The Upper Rhine Valley or 'Rheintalgraben', which literally means 'Rhine Valley rift', originates from the subsidence of the area between the Vosges and the Black Forest as shown in Fig. 2. This must have taken place between the Eocene and the early Oligocene. The valley rift dropped to depths of 7900 to 9200 ft below the adjacent area, not by a single fault but by step faulting. The lateral boundaries are formed on the eastern side by the granite and gneiss of the Black Forest and of the Odenwald and on the western side by the granite of the Vosges and the sandstone of the Palatine Forest. In the so-called 'fore-land

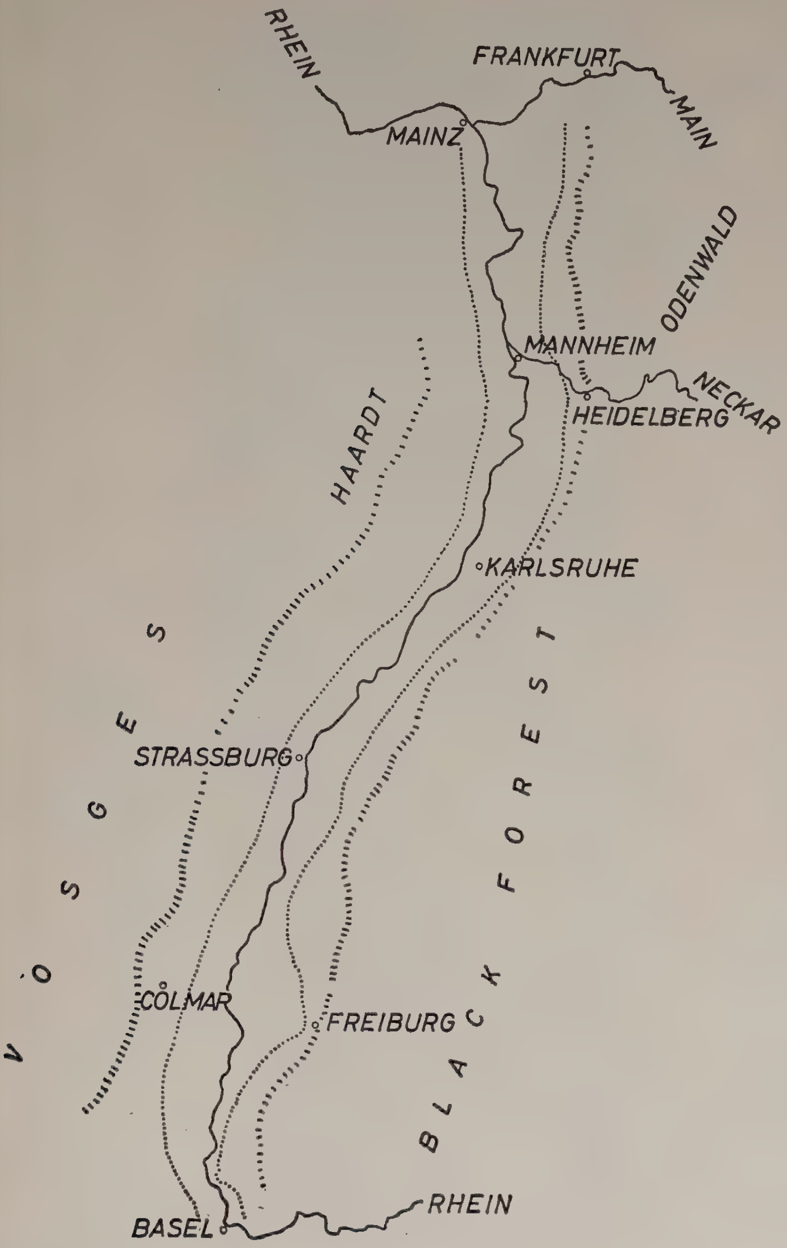
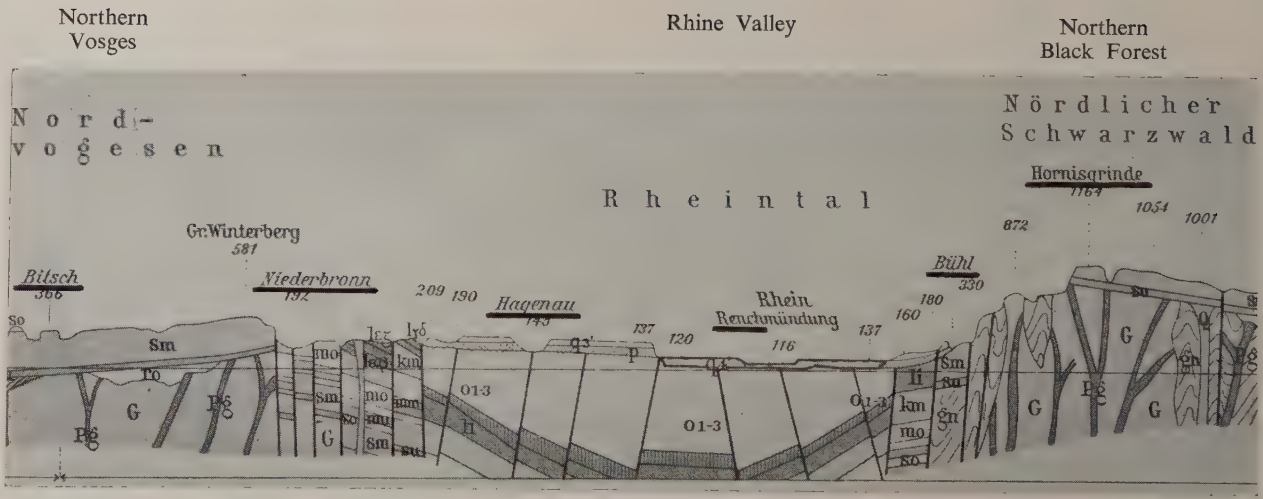


Fig. 1.—Map of the Upper Rhine Valley (Rhine Valley rift).



q4′ Schotter der Würmeiszeit—gravels of Würm Glaciation (main sand and gravel deposit)
q3′ Schotter der Risseiszeit—gravels of the Riss Glaciation
p Pliocäne Schotter—Pliocene gravels
01–3 Oligocäne untere Molasse—Oligocene lower Molasse

Fig. 2.—Cross-section of the Rhine Valley rift.

zone' section, downthrows formed the transition from these areas to the Rhine Valley rift.

During the sinking period the mid-Oligocene sea penetrated into the valley and partly filled it with marine deposits. The geological formations confronting the dredging operation comprise deposits of the Holocene period and gravels of the Würm Glaciation. Gravels of the Riss Glaciation lie on top of Pliocene gravels and sands on both sides of the area which is being exploited. At depths of 260 ft Molasse strata of the Oligocene period are to be found below the alluvials and the gravel beds. Thus dredging has to take place within the layers of the Holocene and Pleistocene period.

Another factor of importance is that owing to the filling of the rift valley from the south very coarse material with pebbles up to 12 in. across is found near Basle, while the sediment becomes gradually finer towards the north, until a very fine sand below $\frac{1}{8}$ in. predominates over the stretch between Mannheim and Mainz.

DEMAND FOR SAND AND GRAVEL

The consumption of sand and gravel up to 1939 was much lower than it is today. The difference in the development before and after the war may be shown by the fact that consumption rose from 1950, which corresponds to the pre-war standard, with an index of 100, to a figure of 413 in 1960.

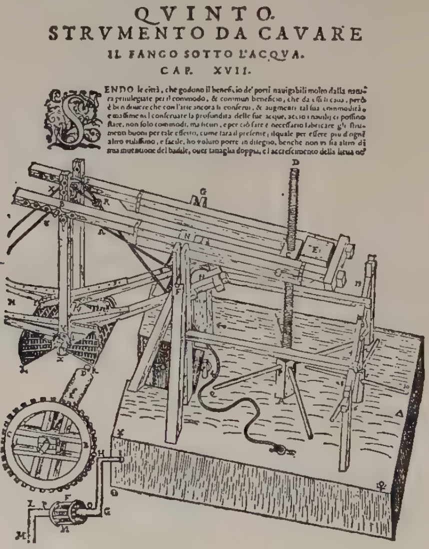
During the period before the war the traditional excavating equipment, such as bucket dredges floating as well as mounted on rails or tracks and scrapers, was adequate. The maximum depth of working under water varied between 7 and 15 yd. Suction pumps and tower-mounted scrapers were also used to some extent, as well as face shovels and draglines for dry working.

The great and increasing demand for sand and gravel for use during the period of post-war reconstruction, and to catch up on the lack of construction pre-war after the beginning of rearmament in 1936, led to a scarcity and to an increase in prices for the purchase or rent of land. These circumstances compelled owners of gravel pits to develop methods of extracting the raw material to be found at greater depths, the scope of these deposits having already been indicated in the geological description given above. The necessary co-operation and technical support was provided by two manufacturers of cranes and material handling equipment and so, some ten years ago, the development of the 'Schwimmtiefgreifer'—the floating deep grab dredge—began in the Upper Rhine Valley.

Today more than 50 per cent of the sand and gravel produced in the Rhine Valley is being extracted by grab dredges. Their use has spread from this area to other parts of Germany and abroad; equipment of this type was introduced into the United Kingdom in 1962.

CHOICE OF DREDGING EQUIPMENT

At this point it is appropriate to refer to the only other equipment suitable for operating at depths similar to the grab dredge, namely the suction pump. Such pumps are also in use in Germany and there are examples in the Upper Rhine Valley, but it appears that the costs for energy and spare parts in Germany are much higher than in the U.S.A., for example, where large suction pumps seem to be the main solution to the problem of extracting sand and gravel from great depths under water.



QVINTO.
STRVMENTO DA CAVARE
IL FANGO SOTTO L'ACQVA.
CAP. XVII.

Fig. 3.—Floating grab dredge of Faustus Verantius (1591).

The idea of grab dredging existed some centuries ago, as illustrated by Fig. 3. The main obstacle against its development was probably the lack of a suitable prime mover to produce the required power. After the invention of the steam engine cranes on pontoons, as well as, later on, mobile grabs or draglines with diesel engines also on pontoons, were developed for excavation operations and the deepening or the cleaning of canals, waterways and harbours.

DESIGN OF A FLOATING GRAB DREDGE

A modern floating grab dredge consists of the following main sections: the pontoon substructure, the grab, the supporting structure for the grab and partially for the winch unit, the grab winch unit with

TABLE I.—*Operating details of grab according to capacity* (Average figures from two different manufacturers)

	Output per capacity of grab, yd³ (based on 50 min working time per hour and 90% filling of the grab)						
	1 m³ (= 1·3 yd³)	1·5 m³ (= 2 yd³)	2 m³ (= 2·6 yd³)	2·5 m³ (= 3·3 yd³)	3·2 m³ (= 4·2 yd³)	4 m³ (= 5·2 yd³)	6 m³ (= 7·8 yd³)
Working depth, yd							
11	59	85	118	144	177	222	347
16	52	72	105	118	150	196	288
22	39	59	92	105	134	170	262
33	33	52	72	85	105	137	203
44						118	170
Two hoist motors, kW, each:	25	30	40	50	60	75	125
Hoisting speed of winch, yd/min:	49	49	66	66	66	66	82
Size of dewatering screen:	8·2 × 2·6 ft	9·8 × 3·3 ft	11·5 × 3·3 ft	13·1 × 4·9 ft	16·4 × 4·9 ft	16·4 × 6·6 ft	16·4×6·6(2)
No. of decks:	1½	1½	2	2	2	2	2

the gears and the hoist motors. The dimensions of the pontoons depend on the capacity of the grab and the eccentricity of operation. This substructure may have a dredging well through which the grab can be lowered and raised. For this purpose the single pontoons are arranged in the shape of a rectangle. Sometimes, for a special type of supporting structure, two separate pontoons are arranged parallel to one another and joined together by the superstructure.

The grab is based on the orthodox design of this kind of equipment, but specially adapted with interlocking teeth for the purpose of dredging sand and gravel. The capacity in yd^3 of the grabs increased with their development from 1·3 to 2·0; 2·6; 3·0; 3·4; 4·2; 5·2; 6·5. Grabs are now available with a capacity of 7·8 yd^3—corresponding to a hoisting capacity of 20 tons.

The supporting structure at first consisted of a luffing jib arrangement which had to be tilted forward to discharge into a bin or over a chute as shown in Figs. 4 and 5. With the development of grabs with greater capacities and higher weights the luffing jib was replaced by an overhead gantry system on which the winch unit travels. The original type of luffing jib has the advantage of a lower centre of gravity with the mechanical part being located on deck. The gantry type has the advantage that it can cover a larger area of deposit under water, but needs an

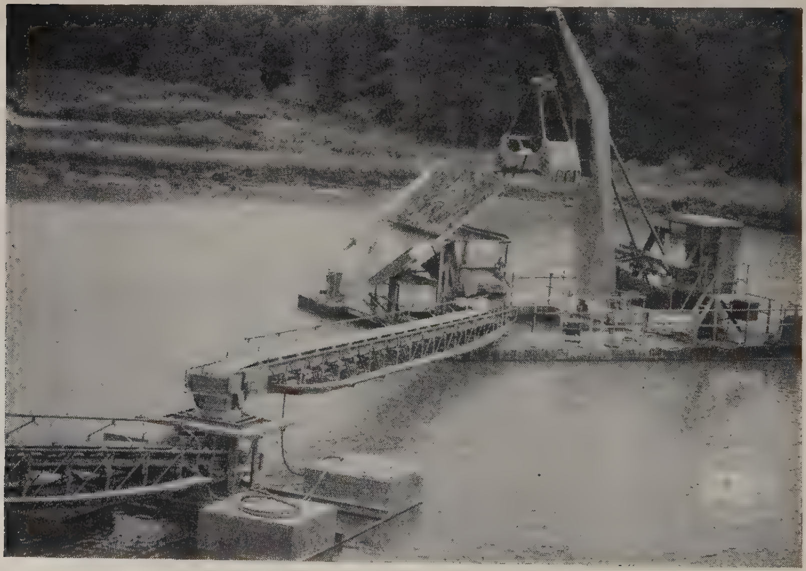


Fig. 4.—Dredge with 1·3-yd^3 grab and luffing jib working over grizzly to a floating conveyor.

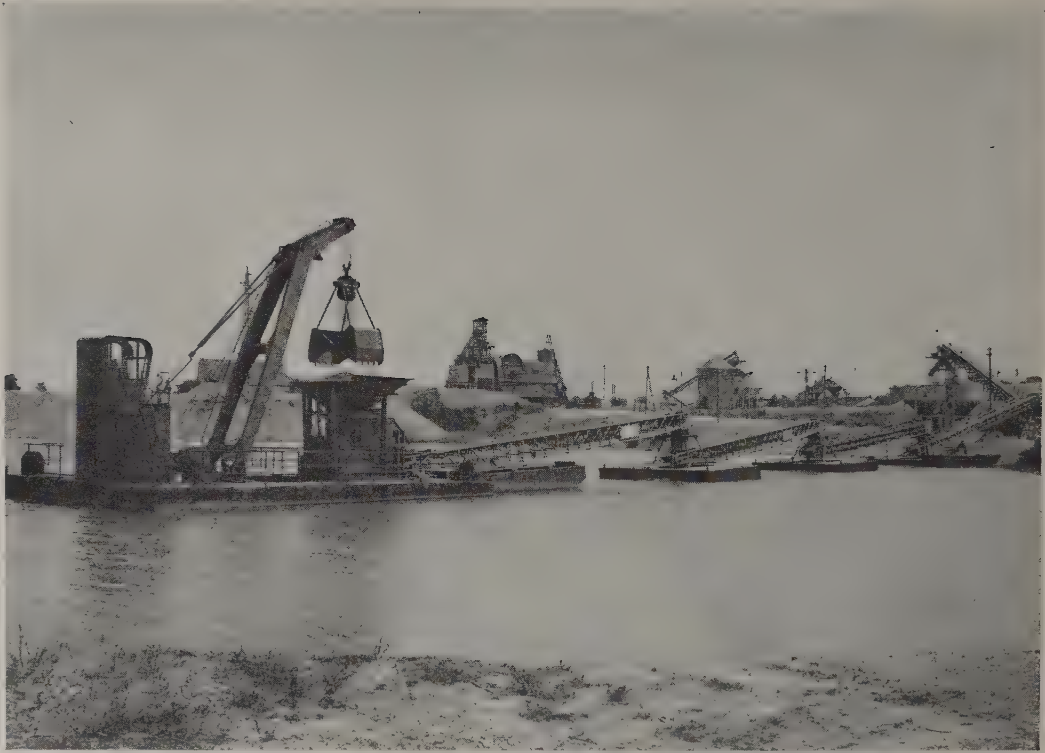


Fig. 5.—Plant with 2·6-yd³ dredge and floating conveyor system.

elevated position for the winches, the mechanical and electrical control system and the operator (see Fig. 6). A later development is shown in Fig. 7. A portal frame, which is based on two parallel but separated pontoons, allows the passage of barges to be loaded by the grab, the winch unit of which is located on the portal. Another construction involves the placing of a slewing crane on a pontoon: this also has the advantage of a comparatively large area of operation under water and is particularly suitable for smaller depths of deposit (Fig. 8).

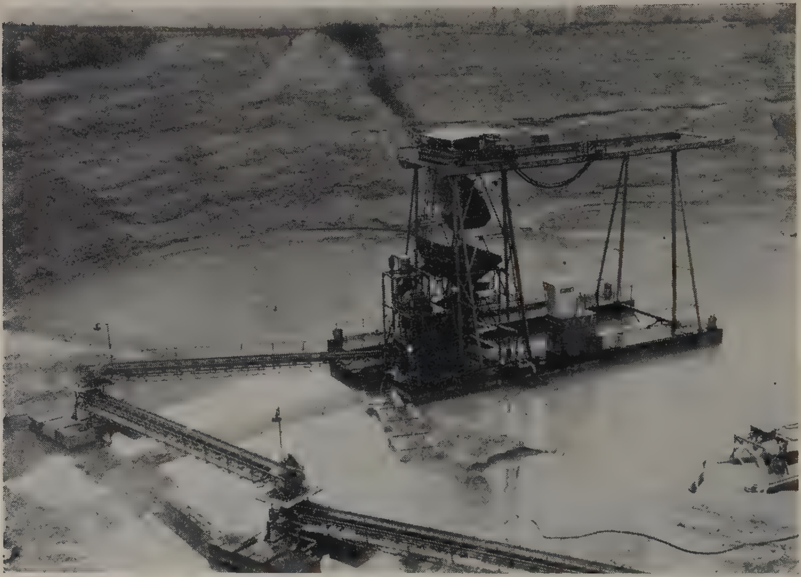


Fig. 6.—Dredge with overhead gantry system and dewatering screen between the bin and the conveyor.

The special winch unit system, with reduction gears and motors, is a vital part of the dredge design in its relation to capacity. The desire to dredge to greater depths, down to 45 yd, led to higher hoisting speeds (increasing from 49 to 66 and now to 82 yd/min) in order to minimize the unproductive time in the dredging cycle. The hoisting system consists of two independent but identical winches, each comprising a grooved rope drum, a fully enclosed helical gear in roller bearings running in an oil bath, coupling, brake and slip-ring motor, each winch catering for half the hoisting capacity of the dredge. The latest types have the two halves of the winch unit in separate sections, so facilitating the removal of one half in case of repair. This arrangement is shown in Fig. 9. Over each drum one rope passes to the grab. If both motors are turning in the same direction at equal speed the grab is lowered or

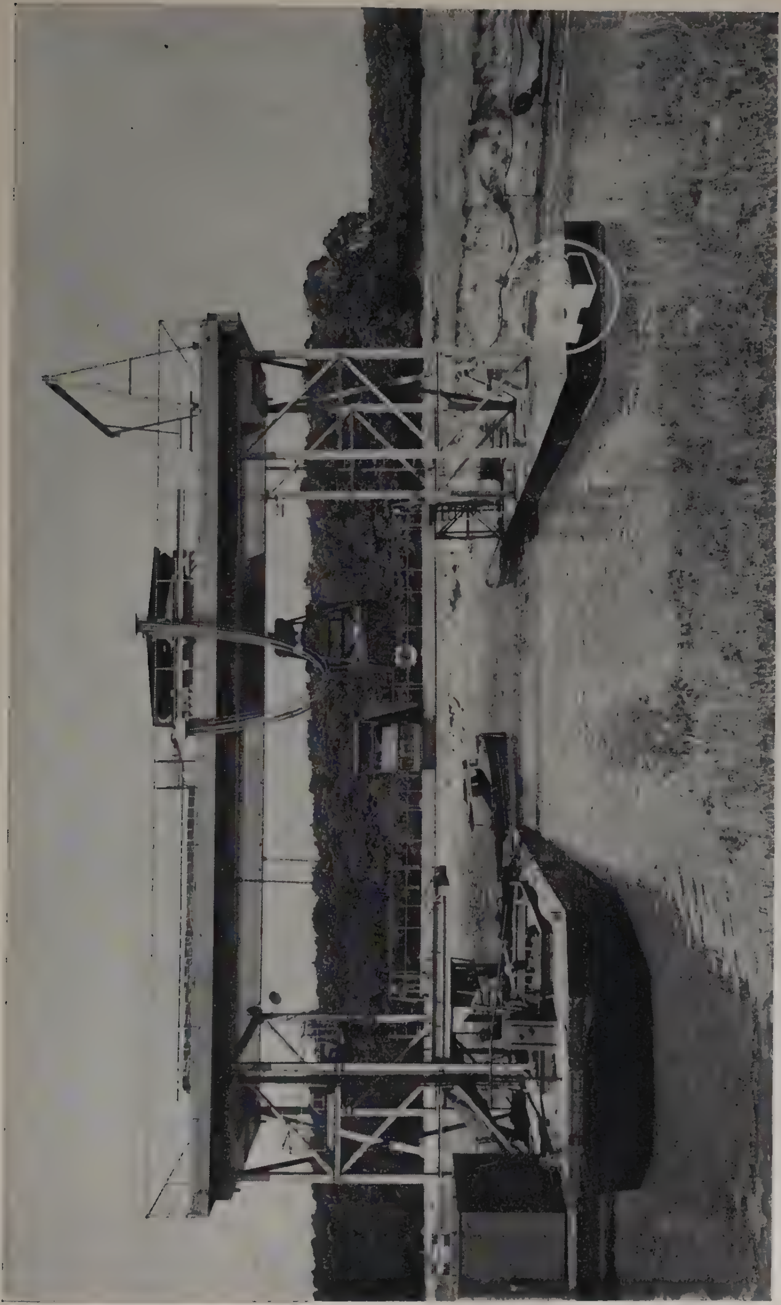


Fig. 7.—5·2-yd^3 dredge using the portal system with bottom-dumping barge.

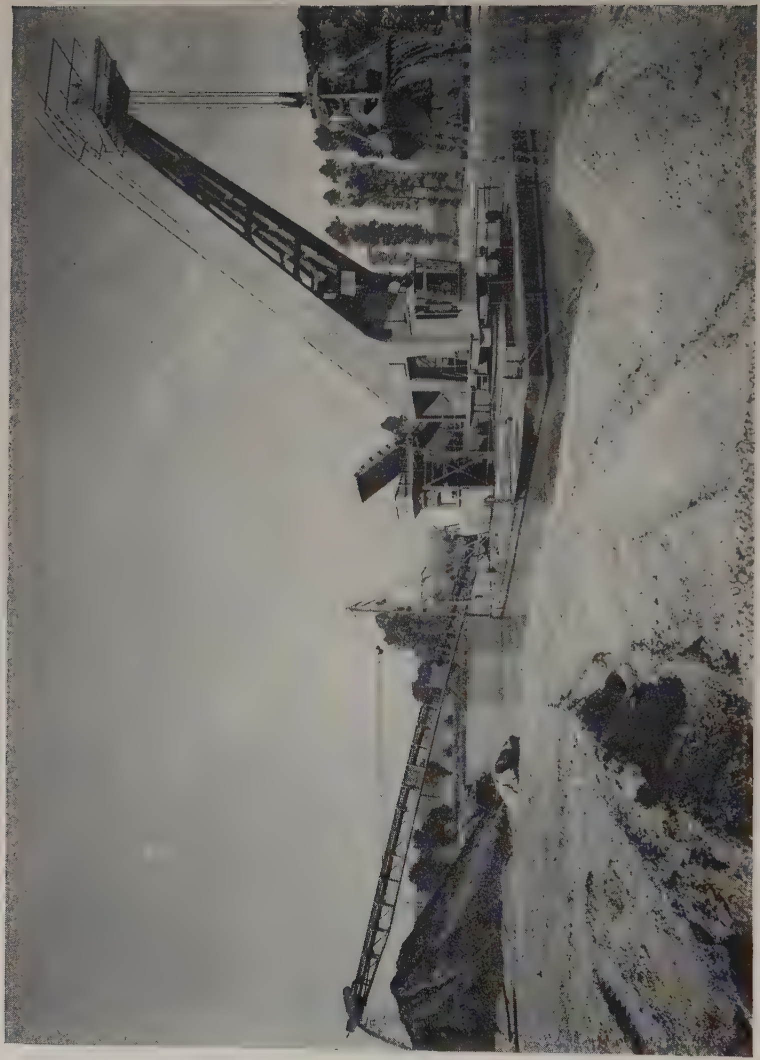


Fig. 8.—Dredge of the slewing crane type unloading over a fixed conveyor on land.

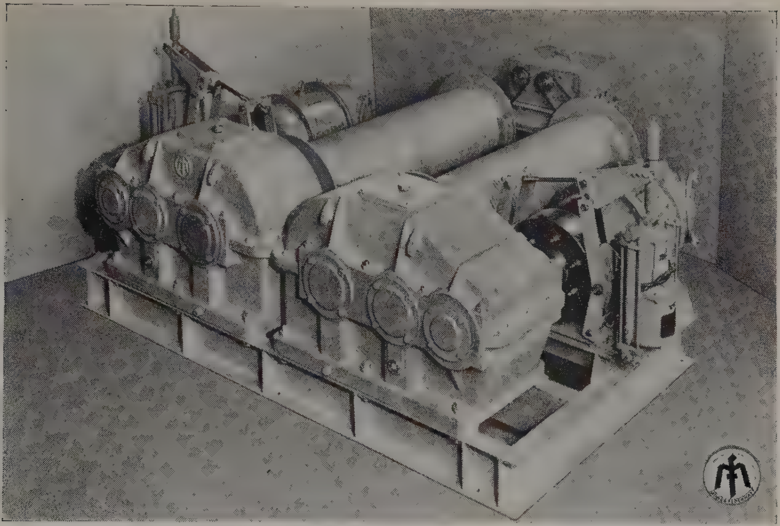


Fig. 9.—Modern winch unit with the covers removed showing hoist motors and rope drums.

raised, the latter action using the full hoisting capacity of the unit. If the closing motor and its winch is working alone, or with a speed differing from the other motor, the grab is opening or closing.

These functions have to be accurately selected by an electrical control system, which also regulates the closing action of the grab under water. Limit switches are used to establish the depth of operation and also the highest point to which the grab is elevated. All the operations nowadays can be automated in order to maximize the capacity, relieve the operator and guarantee an exact depth if required by legal prescriptions or geological conditions. The means for this exact control comprises a very modern electrical system under continual development by the manufacturers.

ASSOCIATED TRANSPORT SYSTEMS

The development of the floating grab dredge to capacities of up to 350 yd^3/h from depths of 11 yd led inevitably to the problem of providing a system of transport corresponding to these capacities. The first rather primitive grab dredges discharged over chutes to ordinary open barges. The unloading of such barges in the traditional way by slewing cranes became generally more and more expensive and unproductive because of the rising cost of the manual labour involved in unloading the last 10 to 20 per cent of sand and gravel from the bottom of the barges. The solution of this problem was found at first by the use of floating conveyor systems connecting the dredge to fixed land conveyors; later, bottom-dumping barges were also employed. The advantages of the use of conveyors are well known and are proved by the extension of this system of transport all over the world during the last 30 years. But some problems had to be solved for the direct transport of sand and gravel won from the water. There is first of all the difference of elevation between the water and the land where the surge pile of sand and gravel or the washing and screening plant are located. Sand and gravel directly delivered from the grab into a bin contains a high proportion of water which limits the inclination of the conveyors leading to the shore. Therefore a dewatering screen is installed between the bin into which the grab is unloading and the conveyor loading point, as shown in Fig. 6. This reduces the proportion of water in the sand and gravel to a percentage which allows the use of the full capacity of the conveyors for slopes up to 15°. The installation of such dewatering screens also allows control of the raw material in thin layers, thus facilitating removal by hand of impurities such as clay, wood, etc.

Fig. 10 shows an example of an installation incorporating a grab dredge, floating and stationary conveyors and washing and screening plant on shore.

The use of conveyors, however, also involves disadvantages apart from the difficulties of starting them in cold weather. Very high investments have to be incurred for conveyors half a mile or more in length with the associated electric lines and transformers. The dredging operation becomes very complicated when depths of 15 yd or more are involved because the embankments carrying the stationary conveyors must have slopes which are flat enough to resist erosion by the action of the water. Their size also makes the removal of these embankments by dredging a complicated procedure.

Development therefore led to the use of bottom-dumping barges which are loaded over chutes or by a dredge of the type shown in Fig. 7 with the winch system working on a portal based on two split parallel pontoons. The material dumped from the barges near the shore is recovered by a slewing crane. Thus a surge pile of material can be held available to bridge interruptions in the dredging or transport operations. The effective range of the ordinary slewing crane being rather restricted, a stock considerably larger can be held by the use of a suction pump or, if this type of equipment is not desired for the reasons already mentioned, a second floating dredge may be used, the grab of which can be limited to a smaller dimension if the working depth is restricted to 10–20 yd.

Another development of the floating dredge consists of a system which does not deliver and transport the raw sand and gravel, but incorporates a plant for washing, screening and sometimes also crushing the raw

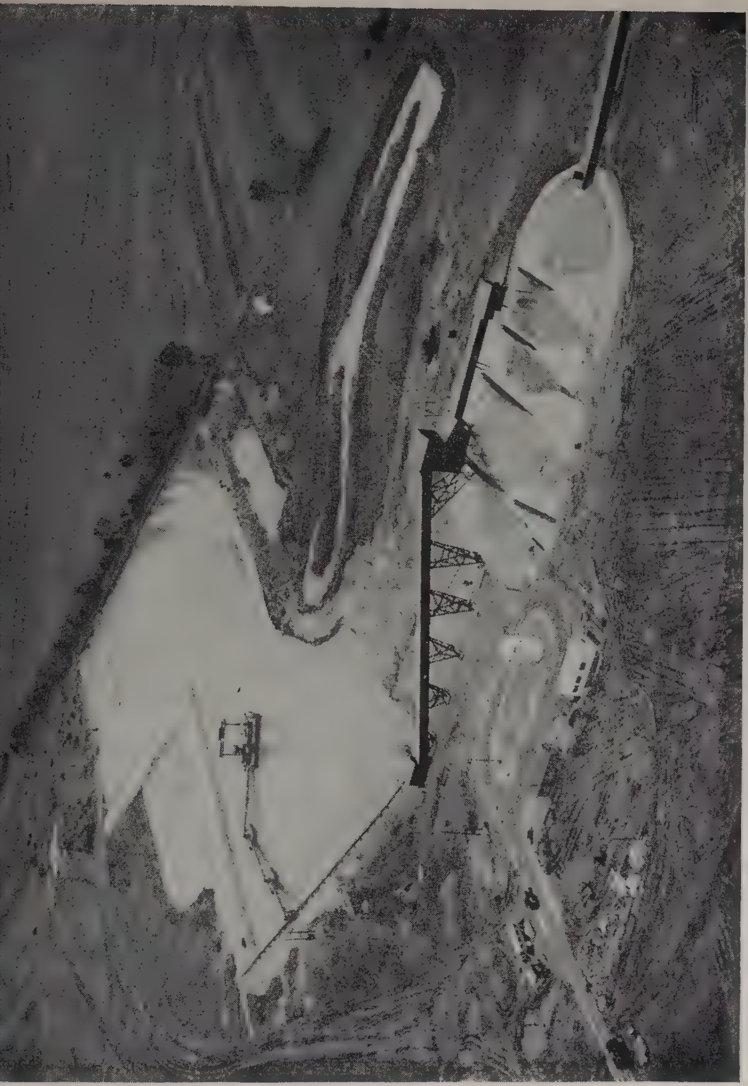


Fig. 10.—Washing and screening plant of Karl Epple at Liedolsheim (downstream of Karlsruhe) with grab dredge, floating and stationary conveyor system.

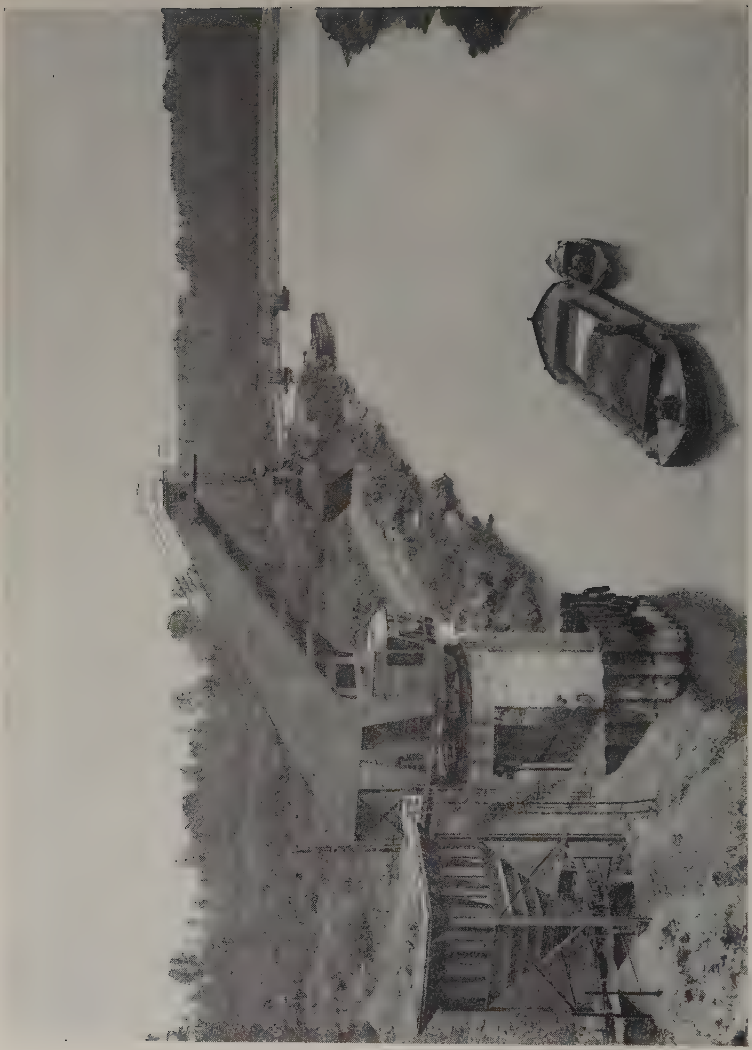


Fig. 11.—Installation of Kieswerk Illingen (upstream of Karlsruhe) with a grab dredge of the portal type, a bottom-dumping barge and a slewing crane unloading over a bin to a conveyor.

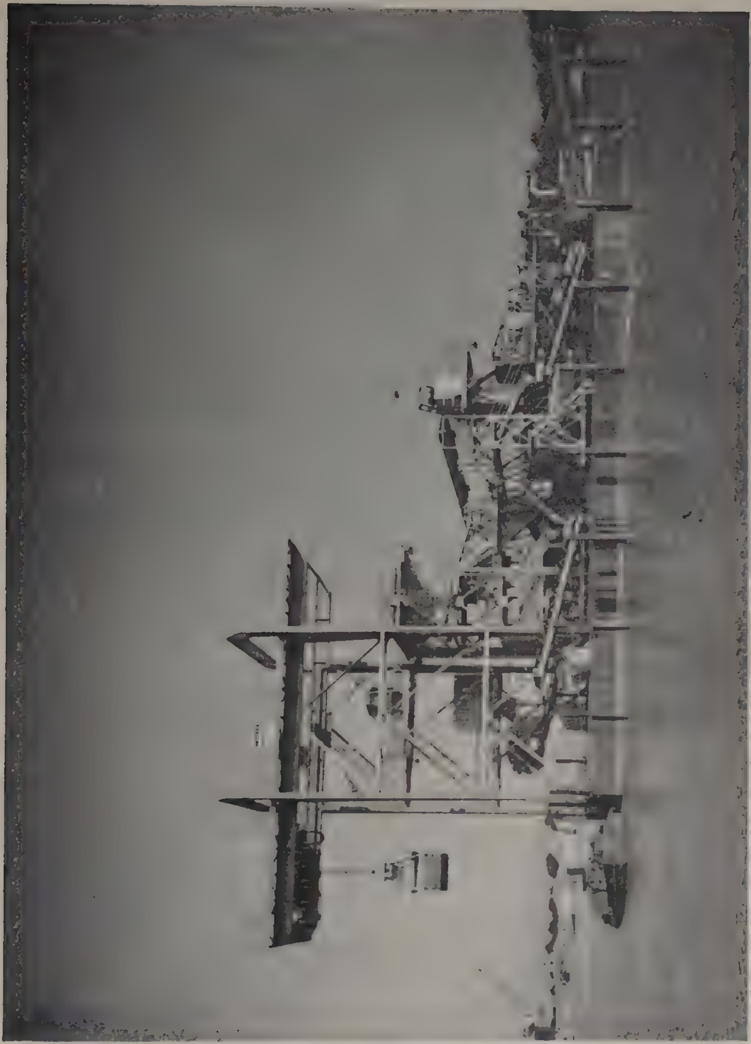


Fig. 12.—A Meichle and Mohr installation on the Lake of Constance with 5·2-yd^3 dredge with integral washing and screening plant.

material. This method avoids unproductive transportation. On the other hand, it is impossible to maintain stocks and the single grades of the produced material must be very exactly disposed of, otherwise one or other grade must be put back in the water. Such floating grab dredges incorporating a processing plant are shown in Figs. 12 and 13.

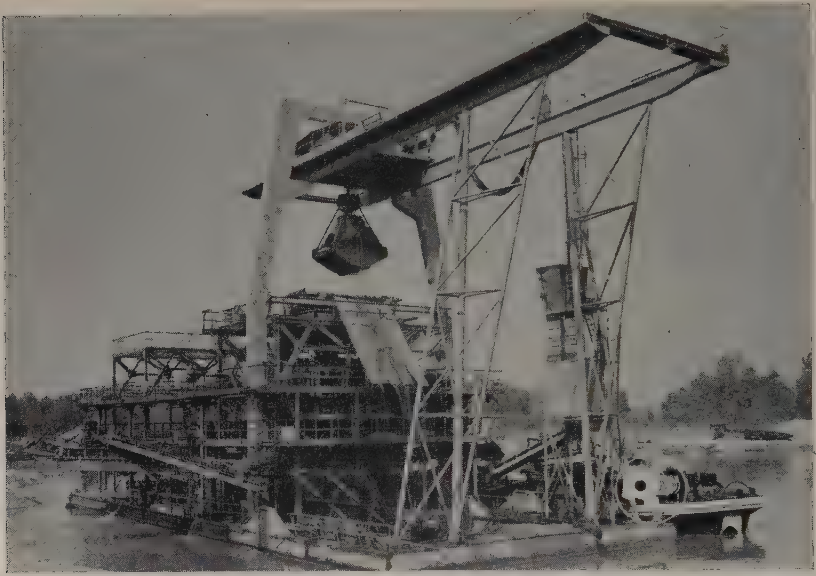


Fig. 13.—Similar installation of Gebr. Willersinn near Munchhausen (Alsatia) with 7·8-yd^3 dredge.

Anchoring and mooring

Anchoring and mooring of the floating grab dredges take place in the same way as for traditional floating bucket dredges, i.e. by cables over cable winches at the four corners of the dredging unit. The cable winches can be driven by hand or by electric motors, in the latter case controlled from the operator of the grab winch unit. In order to facilitate the approach and mooring of barges the cables can be passed over pulleys fixed at elevated points of the superstructure of the dredge. The cables are affixed to ground pegs on shore or to anchors in the water.

The present situation to which the developments described above have led shows a trend towards: (1) grabs with a capacity of 8 yd^3; (2) the overhead gantry system, sometimes with an open portal; and (3) winches and grab motors for hoisting speeds of 66–82 yd/min.

MAINTENANCE AND COSTS

The wear on floating grab dredges is concentrated on the ropes, but the teeth and the bearings of the grab and the bearings of the winches are also exposed to wear. The indispensable minimum of major spare parts is: one grab, one winch drum, one hoist motor and a certain number of electrical components. In general, and for German conditions, it may be said that the repair costs for a grab dredge for the same kind of material, working depth and output are lower than for a bucket dredge or for a suction pump.

Table II shows figures for the costs per yd³ of material dredged for different types of floating grab dredges and for different working depths and capacities.

TABLE II.—*Cost prices for floating grab dredging*
(Omitting costs for dewatering screen)

Capacity of grab:	1·5 m³ (= 2·0 yd³)	2·5 m³ (= 3·3 yd³)	4 m³ (= 5·2 yd³)	6 m³ (= 7·8 yd³)
Approximate price of dredge with pontoons including assembly	£12 200	£15 200	£25 900	£38 000
Depreciation within 10 years at 0·83%/month	£102	£127	£215	£316
For 200 hours/month, per hour	10·2 s.	12·7 s.	21·5 s.	31·6 s.
Repair, 133%	13·6 s.	16·8 s.	28·5 s.	42·2 s.
(Motor output	(60 kW	(100 kW	(150 kW	(250 kW
Current consumption)	40 kWh)	67 kWh)	100 kWh)	167 kWh)
at 2·6 d./kWh/h	8·6 s.	14·3 s.	21·4 s.	36·1 s.
Lubricants, 15%	1·3 s.	2·1 s.	3·2 s.	5·4 s.
Wages for 2 men	12·5 s.	12·5 s.	12·5 s.	12·5 s.
Social costs, 50%	6·2 s.	6·2 s.	6·2 s.	6·2 s.
Cost prices per hour	52·4 s.	64·6 s.	93·3 s.	134·0 s.
Output per hour: cost per yd³				
for 11 yd depth	85 yd³: 7 4 d.	144 yd³: 5·4 d.	222 yd³: 5·0 d.	347 yd³: 4·6 d.
16 ,, ,,	72 yd³: 8·7 d.	121 yd³: 6·6 d.	194 yd³: 5·7 d.	292 yd³: 5·6 d.
22 ,, ,,	60 yd³: 10·4 d.	108 yd³: 7·4 d.	173 yd³: 6·6 d.	259 yd³: 6·1 d.
33 ,, ,,	50 yd³: 1s. 8 d.	85 yd³: 9·1 d.	141 yd³: 8·2 d.	206 yd³: 7·9 d.
44 ,, ,,			118 yd³: 9·5 d.	173 yd³: 9·5 d.

Acknowledgements.—The author is indebted to Professor Dr. J. Bartz (Freiburg) and Professor Dr. M. Frank (Stuttgart) of the Geologisches Landesamt Baden-Württemberg for valuable information regarding the geological aspects; to the two manufacturing firms, Mohr & Federhoff A.G. and A. Ridinger, both at Mannheim, for providing data and photographs.

621.879.4:622.271.1

Paper 3

Some Developments in Connection with the Use of Suction Dredges for Mining Alluvial Deposits

J. C. C. HILL, Affiliate I.M.M.

Independent Consultant; formerly Chief Engineer, Dredger Division,
Acrow (Engineers), Ltd., London

SYNOPSIS

The selection of excavating equipment for alluvial mining is discussed and the use of suction dredges for the development of certain deposits is outlined. Reference is made to earlier and less successful dredges.

A brief description is given of some modern cutter suction-dredging operations for overburden removal and some smaller units.

Similar units are considered as main dredging equipment for suitable areas, equipped with either self-contained washing plants or pipelines for transporting the mineral slurry to separate plants. Detailed illustrations are included of dredges and treatment plant for excavating diamondiferous gravel from a West African river bed.

The possibility of using such equipment for prospecting is considered. Methods of applying small suction dredges, especially those employing the jet-venturi principle, to the cleaning up of mineral-rich ground from between hard bedrock pinnacles are discussed with particular reference to their use in conjunction with special dredging units operated by divers using compressed-air equipment.

In this paper some recent mining operations are described that employ specialized suction-dredging equipment for excavating mineral-bearing gravel. Although the operations detailed concern mainly diamond-mining operations, the general observations and the experience gained from these undertakings can be applied to general mineral dredging with particular reference to suction dredging.

Suction-dredging equipment has not always received the consideration it deserves when selecting excavating equipment for the development of new alluvial mining areas. Furthermore, in many parts of the world where mineral-bearing areas occur the feasibility of working rivers is often considered only with regard to the use of bucket-dredges, river diversion and dry working. Rivers cutting across mineral-bearing ground sometimes offer extremely rich sources of mineral, their action in many cases being similar to that of a sluice in which the light gangue is washed away and the heavy mineral concentrated. They are in many instances unsuitable for working by the three methods mentioned, but can often be successfully developed by the use of suction dredges.

The dredging of offshore mineral deposits is also receiving increasing attention from mining companies, one noteworthy advantage of suction-dredging equipment in such applications being the ability to dredge to considerable depths. This, coupled with the relatively low cost of straight suction-dredging equipment, will without doubt allow the further development of mineral deposits that are known to exist on the sea bed.[1]

To some extent the apparent lack of interest in this type of equipment may be due to the recollections of some of the ill-fated suction dredges installed earlier in this century for the excavation of minerals, particularly some of those operating on Malayan tin-bearing ground.[2] The total failure of some of these machines was remarkable, but understandable in view of the lack of experience of this type of application.

The design of suction dredges, as with most plant, must be undertaken from basic concepts, each of the components forming an integral part of the complete dredging unit, and each component being designed to operate in conjunction with and complementary to each other item. Thus the pump capacity, winch speed, cutter, controls and processing plant must be designed as a whole to form a complete and efficient unit to suit the deposit, operating conditions and production required.

Of particular interest in More's paper[2] are his remarks concerning the ability of a suction dredge to raise heavy mineral particles. This ability has been questioned by engineers connected with tin-mining operations, but theoretical considerations and the record of successful operations undertaken in more recent years by suction dredges for the recovery of heavy minerals in many parts of the world contradict those opinions.

The familiar gravel pump has the same operation in theory as a dredge pump, the main difference in practice being that on the one hand the gravel pump is stationary and mounted on land, the material being brought to the suction by sluices, while in the case of the dredge pump the suction is kept in contact with the material by moving the dredge. Provided the suction orifice is sufficiently close to the material to allow the effect of the surrounding water moving into the pipe to act on the particles, and provided the velocity of this water is great enough to give sufficient carrying capacity, i.e. to be substantially higher than the settling velocity of the largest particle to be raised, the particles must enter the suction pipe. Suction velocity is, of course, solely related to the quantity of water being drawn into the pump and the size of the suction pipe. These conditions are in turn related to the suction characteristics of the pump and its efficiency; also to the dredging depth. The shortcomings of the earlier dredges most likely resulted from too low a suction velocity.

This was also apparently aggravated by the use of short sluices for separating the tin. These sluices, discharging their tailings close to the stern of the dredges, gradually reduced the size of the paddock to impractical dimensions for manoeuvring the dredges, and also caused the tailings sand to build up against the foot of the working face where it was again drawn into the pump, so giving rise to the impression that the heavy mineral was being left behind.

However, the important question in all suction-dredging projects is not whether the mineral will be picked up by the dredge, but whether the physical characteristics of the ground to be mined are suitable or, by the use of cutters or disintegrating high-pressure water jets, can be made suitable for the action of the water entering the suction pipe to pick up the particles.

Some examples are given below of recent successful suction-dredging operations undertaken in connection with the exploitation of opencast deposits. The first two dredge descriptions are of those of large cutter suction dredges used for large-scale overburden stripping operations, while the last two accounts are of smaller suction dredges used for the excavation of mineral-bearing ground.

OVERBURDEN STRIPPING OPERATIONS

Notable among the overburden stripping operations were those of the Caland Ore Co., Ltd., in Canada.[3] This project called for the removal of 161 000 000 yd^3 of varved clays and silt from a deposit of iron ore. The maximum depth of overburden to be removed was 400 ft and approximately 12 miles of 36-in and 42-in diameter discharge pipe were used.

Of special interest in this operation was the method for controlling

TABLE I.—*Main specifications of Caland dredges*

Hull dimensions:	176 ft × 50 ft by 14-ft draught
Total hp on discharge pipeline:	30 000
hp on each of the three pump motors:	10 000
Cutter hp:	1000
Cutter rev/min:	18
Maximum depth of cutter:	50 ft
Pump impeller diameter:	92–110 in. (weight 7–10 t)
Impeller clearance:	21 in.
Pump casing thickness:	6–8 in. (weight 40 t)
Floating discharge line diameter:	36 in.
Floating discharge line length:	2 miles
Shore pipeline length:	9½ miles of 40-in and 42-in pipe
Maximum static head between the discharge area and the dredging paddock:	777 ft
Power:	115 000 V to sub-station; from there to dredges and boosters at 13 800 V
Pipeline velocities:	10–21 ft/s
Percentage solids by volume:	6·5 to 21
Output:	800–6850 yd^3/h
Average output per hour:	3555 yd^3/h (pumping time only)

the runoff water so that as dredging proceeded the level of the lake in which dredging was taking place could be lowered in readiness for the next cut. In this way the dredging equipment, with a maximum operating dredging depth of 50 ft below the water-level, was able to excavate 400 ft of material by stages.

The first of these large dredges was in operation by 1955 and the overburden removal was completed by 1960. Two identical cutter suction dredges were employed, the 'Clarence B. Randall' and the 'Joseph L. Block'. The design and size of these dredges are of interest, the main specifications and dimensions being given in Table I.

As a further example of this type of dredge, also used for overburden stripping, that employed in Scotland for an opencast coal operation is of interest. This machine was equipped with two 7½-ft diameter pumps, each driven by a 1520-hp motor. The pumps delivered an average of 20 700 gal/min of a mixture of water and solids at a total head of 70 ft over a mile of 26-in delivery pipe. The material dredged is conveyed by the pipeline to storage areas which are divided into compartments to allow the finer particles to settle out, the total storage capacity being in the region of 4 000 000 yd^3.

The dredge used for this operation is capable of handling an average of 900 yd^3/h of solids. During continuous working, except for weekends, the output of the dredge was 4 382 000 yd^3 in 49 working weeks, an average of some 90 000 yd^3/week.

These are examples of the more conventional type of cutter suction dredge. Smaller units, specially designed to meet specific local conditions, have also been installed recently for the excavation of mineral-bearing gravel from river beds. Instead of the mechanical rotating cutter head these use high-pressure water jets for disintegrating the material.

MINERAL SUCTION DREDGING

Suction dredges for elevating mineral-bearing deposits from below water-level may incorporate the treatment plant on the same pontoon or combine with a separate treatment plant fed from the dredge through a floating pipeline.

The dredge shown in Fig. 1 has a flotation constructed from pin-connected standard pontoon sections on which a complete diamond recovery plant is installed with a jet-venturi-type dredge pump.

The dredging equipment comprises a jet-venturi dredging ladder for a depth of 50 ft, powered by a 110-hp diesel engine. The discharge of dredged material is elevated by pipe 25 ft above water-level to a rubber-lined boiling box which feeds a double-deck vibrating screen. The oversize from this screen is discharged into a chute which passes the material to a barge moored alongside. The lower deck of the screen

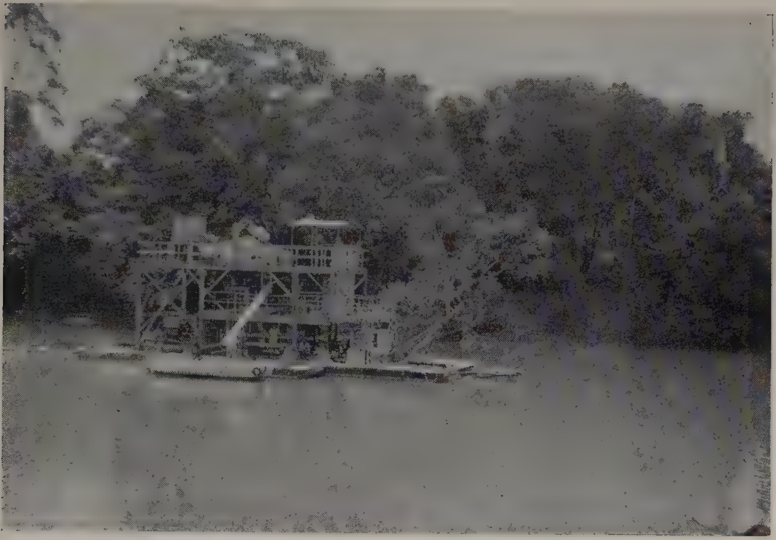


Fig. 1.—General view of a jet-venturi-type dredge equipped with a combined mineral-processing plant.

separates the remaining material into two fractions, the oversize being fed to the jigs where the diamonds are recovered, while the undersize, joining the jig tailings, is discharged by a solids handling pump through a tailings pipeline at the stern of the dredge.

The controls of the dredge are on the third deck on a control platform. This cabin houses the hydraulic gauges, together with the control levers for the hydraulic winch employed for moving the dredge and also for raising and lowering the dredging ladder. Remote control of the high-pressure agitator water is also located there for controlling the high-pressure water-nozzles used for disintegrating compacted material.

The main dimensions of the combined dredge and treatment plant are: beam, 38 ft; pontoon length, 60 ft; overall length, 80 ft; pontoon depth, 4 ft; and overall height, 26 ft.

This dredge is being used for excavating diamondiferous gravel from a West African river bed. For ease of transportation and erection a sectional pin-connected construction is employed with a maximum weight of any component of 3 tons.

An approximate percentage analysis of the material being dredged is: sand, 50; sandy gravel, 33; gravel, 10; bedrock, 5; and timber and vegetation, 2 per cent.

The approximate capacity of the washing plant on the dredge illustrated in Fig. 2 is 25 yd^3/h, its output being estimated at 23·25 yd^3/h.

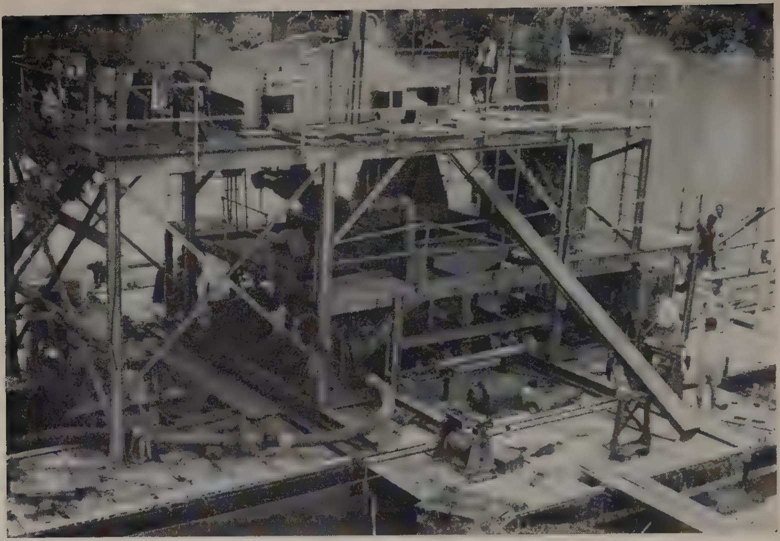


Fig. 2.—Arrangement of dredge-mounted treatment plant.

The total estimated output dredged by this machine up to June 1963 was 28 328 yd^3.

The dredge shown in Fig. 3 is designed for operation separately from

Fig. 3.—View showing dredge being used for excavating mineral-bearing gravel.

the washing plant, the mineral-bearing gravel being pumped through a floating pipeline to the treatment plant on shore.

As with the dredge described previously, the flotation is formed of pin-connected pontoon sections. There are six main pontoon sections, each measuring 15 ft by 6 ft by 4 ft and both the bow and the stern are provided with scow ends to reduce the drag when high water currents occur.

A combined system of pumping equipment is employed comprising a jet-venturi-type pump at the lower end of the suction pipe which feeds, via a stonebox, a high-efficiency centrifugal gravel pump. In this way the dredge can be used to maximum efficiency for deep dredging, the jet-venturi system feeding a high concentration of solids to the centrifugal pump.[4]

The dredge illustrated in Fig. 3 has a maximum dredging depth of 60 ft below water-level, but by the addition of pontoon sections and pipe sections this can be increased to more than 100 ft. To allow for variations in the discharge head two diesels are provided, the first powering the jet-venturi pump and the second, through a V-belt drive, the centrifugal gravel pump. An alternator set provides electrical power for the warping winches and the ladder hoist. Owing to the strong water currents expected in the rivers during the wet season 2-ton electric reversing winches were installed, fitted with $\frac{3}{4}$-in diameter wire ropes.

The maximum weight of the largest component part of this dredge is 3 tons and assembly of the unit from the day of delivery of the first parts to completion of the erection took a total of 28 days without the use of a crane—all the parts being winched into position on timbers. All transportation was undertaken by 3-ton tipping vehicles. A total of 500 European and 800 African man-hours was required for erection.

TABLE II.—*Main specification of jet-venturi/dredge pump dredge*

Dredge hull dimensions:	50 ft × 20 ft by 2-ft draught
Total hp on discharge pipeline:	85
hp on jet venturi pump:	85
Maximum dredging depth (as illustrated in Fig. 3):	60 ft
Impeller diameter:	17 in.
Impeller clearance:	3 in.
Discharge pipeline diameter:	8 in.
Maximum discharge head:	80 ft
Power:	Diesel
Pipeline velocities:	12–23 ft/s
Percentage solids by volume:	4–33
Maximum output achieved:	120 yd^3/h
Average output per hour in sand overburden:	111 yd^3/h

The suction pipework of this dredge is of unusual design to facilitate the excavation of material from narrow and sometimes deep potholes in the river bed. Originally, development work in this connection was along the lines of vertically lowered ladders controlled by winches and high sheerlegs. This method, however, when compared with conventional single-pivoted rigid suction ladders, sacrificed some of the control of the lower section and it was finally decided that an additional set of pivot arms was required, this giving the necessary flexibility without reducing the amount of control over the suction orifice. The final arrangement consisted of a standard single-pivot ladder with the addition of secondary pivots at a distance two-thirds along the suction pipework, allowing the lower section of the ladder to be entered vertically into potholes with the aid of an additional winch.

In addition to the normal dredging controls the operator was also provided with information by echo-sounding equipment, and to afford close control of the feed to the plant a radio-telephone system was installed.

TYPE OF DEPOSIT

The type of deposit is of special importance when considering the use of suction dredges, and the problem of assembling sufficiently comprehensive data on the deposit to allow the correct selection of suction dredge should not be minimized. Owing to their nature some deposits are more efficiently and economically worked by other methods.

Specially designed suction dredges are used for the recovery of diamonds from fast-flowing West African rivers in both operations described here, the first utilizing a jet-venturi-type pump for elevating the dredged material to the treatment plant mounted on board the dredge, and the second unit employing the ‘combined system’ of dredging where the dredged material is elevated to a booster pump on the dredge and thence discharged via a floating line to a treatment plant on shore.

Although the following description is for one area only, the general conditions apply equally well to both, as both rivers are similar in character.

Diamondiferous gravels of West African rivers

Typical of the West Coast of Africa, the area lies in the belt of tropical rain forest and comprises, in the main, gently undulating lateritic surface with a general slope towards the sea of approximately 10 ft per mile. It is completely covered with high bush, making the construction of road communications difficult. All dredging equipment was therefore designed to be broken down into easily handled sections, the heaviest of which did not exceed 3 tons in weight.

The rivers in which the dredges were to operate were both typically tropical, having a period of extreme low water during the dry season followed by sharp increases of flow at the beginning and during the wet season.

During the initial operations under consideration the velocity of the river's flow varied from 0 to 8 knots, although when the rains began higher water velocities were experienced for short periods, it being possible for the river to rise 22 ft overnight and on one occasion a rise of 8 ft was recorded in one hour.

To reduce the danger of the dredge being swept away by these floodwaters a paddock was excavated in the river bank to afford a safe anchorage when the first rains caused the river to rise. The problem remained, however, that in one of the localities very little reliable knowledge was available on the effect of distant rainstorms up river and this could only be gained by experience.

An additional hazard during this time of the first heavy rains comes from timber of considerable size in the river, with large quantities of refuse accumulated during the dry season and swept off the river banks by the high current and rising water. On the first occasion it was estimated that timber and refuse weighing in the region of 30 tons collected on the bows of the dredge, including one tree measuring some 40 ft in length and 4 ft in diameter. On another occasion similar large waterlogged trees floating just below the surface sheared off a suction pipe. As knowledge of the river increased, however, these dangers were to some extent eliminated by deflector wires and by mooring the dredge in a sheltered position when such hazards were likely to occur.

A typical view of the river is shown in Fig. 3. Generally about 200–400 ft wide, the course is approximately straight for distances of 2–5 miles between faults, where the river changes course abruptly for short distances. Rapids occur here, and in the nearby potholes and fissures the highest values were normally found where the natural unevenness of the country rock has resulted in concentration and retention of the heavier minerals.

Before dredging and treatment plant were installed a survey was carried out to ascertain approximately the type of material that would be found in the river, together with information that would lead to a reasonably accurate estimate of grade of ground. This survey was done using pontoon-mounted drills, divers with compressed-air equipment and airlifts, and finally by damming and draining one of the potholes. The most practical method of raising samples was that of divers employing small portable units for elevating the material from the river bed.

For as complete an understanding of the river bed as possible a suitable pothole was dammed off and drained. In this particular instance the nature of the surrounding rock outcrops made only a limited amount of earth dam construction necessary. Draining was begun early in the dry season of 1961, a total of approximately 1 500 000 gal of water being removed by diesel-driven pump. The gravel contained in this hole was removed by hand labour and treated for minerals by hand-operated wooden screening and jigging equipment.

Measuring some 200 ft long and 40 ft deep at its maximum, the pothole contained five distinct strata of sands and gravels. The deposit was overlain by coarse free-running sand, except for a central section where seasonal erosion caused by a circular motion of the flood water had resulted in its removal. In this sand-free area there occurred a layer of large angular rocks which formed a capping on the underlying gravels in which most of the values were found. This gravel layer averaged about 5 ft and contained large angular boulders similar in type and shape to those above, but generally decomposed to such an extent that they were friable.

Where the gravel contacted the granite bedrock this also was decomposed for a considerable depth. In certain areas this deposit of diamondiferous gravel rested on a thin layer of coarse yellow sand free of larger rock particles, and below this two further deposits of relatively fine sand occurred, together with some gravel, separated by a layer of vegetation and timber. Several large boulders in the deepest part of the pothole were removed by winches.

Although the size distribution of the river gravels varied considerably from place to place, it was generally found that some 40–60 per cent could be classified as barren sand, the remaining 60–40 per cent with diamond values being above 2 mm. More recent work has shown that values occur below 2 mm and operations have been modified accordingly.

While considerable deposits of barren sand overlie the older diamondiferous gravel in some areas, in others it is covered with stones and boulders varying in size from a few inches to many feet in diameter; these present a considerable problem in the selection and operation of dredging equipment.

The depth of gravel also varied considerably in different parts of the river, from up to 15 ft deep to possibly only a few inches resting on the bedrock, nevertheless often extremely rich in diamonds. The maximum dredging depth so far experienced is 60 ft.

DESIGN OF MINERAL SUCTION DREDGES

In designing suction-dredging equipment for the excavation of mineral-bearing material, the method of processing the excavated material is an important consideration, and an early decision is necessary on whether the treatment plant is to be mounted on board the dredge as an integral unit or erected separately with a connecting method of feed. Five arrangements are possible:

(1) the dredge constructed complete with the treatment plant mounted on board;
(2) the dredge to feed a separate treatment plant through a pipeline, the plant being erected on shore;
(3) a similar arrangement to that outlined in (2), but with the treatment plant mounted on separate pontoons moored alongside the bank;
(4) the primary treatment of the material carried out on board the dredge and the final treatment undertaken in a separate plant as in (2) or (3); or
(5) the dredge to discharge into barges which are then towed to the treatment plant where the material is grabbed or pumped out and fed to the plant.

The main advantage of having the treatment plant on board the dredge (1) is that the dredgemaster is visually in contact with the feed to the plant, which affords him direct control of production. Other advantages are that this arrangement normally requires less horsepower for the dredging operation and also allows the simplification that all power can be supplied by a single alternator and, using the jet-pump system of pumping, all water power, including that of dredging, from a single centrifugal water pump. Economy of operation is also improved in that all operations are combined in one unit, and security arrangements are easier.

Some disadvantages result from the additional weight of treatment plant, material and water, which requires additional buoyancy and consequently a substantially bigger pontoon. Such a dredge is less suitable for operation in narrow rocky rivers where manoeuvrability is of prime importance. The increase in size also offers greater resistance to any river currents that may occur requiring larger winches and wires.

Furthermore, with this arrangement it is not normally possible to stockpile material; in addition, any mineral lost from the plant cannot be recovered and there may be a problem where river pollution has to be avoided.

The alternatives do not require as large a dredging plant. With separate treatment plants the feed would go by pipeline to the washing plant on shore (2) or on separate pontoons (3). Where rock outcrops, rapids and potholes are common in the area to be worked, the size of the dredge and its manoeuvrability, coupled with the ability to enter narrow sections of the rivers, is of considerable importance.

Certain designs for such an arrangement (Fig. 4) allow the complete dredge to be winched out and transported complete on a low-loader. The ease of moving this type of equipment, coupled with a possible output of 120 yd^3/h, opens up numerous possibilities with regard to small erratic deposits that do not warrant the installation of larger equipment.

Dredges operating separately from the plant allow a more flexible feed arrangement to the processing plant in that excess can be stockpiled and fed at a constant rate. It should be noted that feed from a suction dredge normally varies considerably from moment to moment depending on the ease with which the material enters the suction pipe.

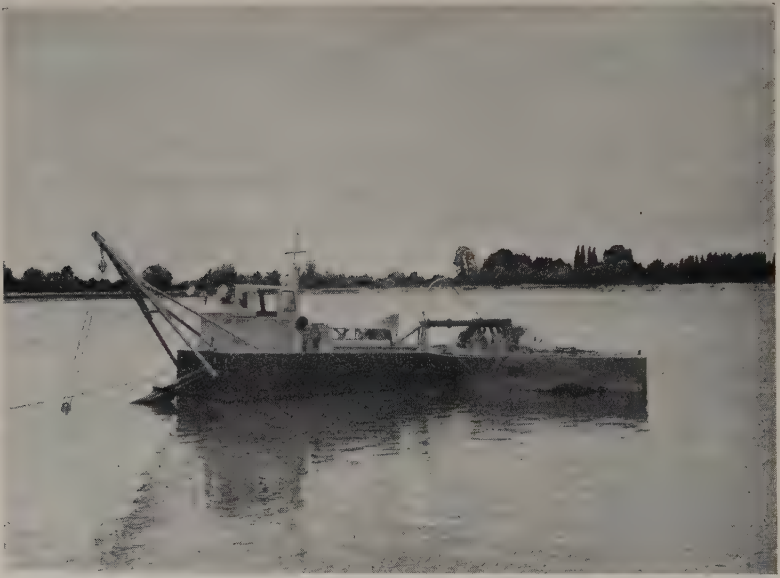


Fig. 4.—Small dredging unit being employed for barge loading. Such a dredge is transported from site to site with the minimum of dismantling.

Where a pipeline is used to transport material from the separate dredge to the treatment plant there may, however, be some difficulties in rivers where there are strong currents and timber and other floating refuse.

The separate washing plant on shore is normally constructed for easy transportation to a new site or, where river conditions allow, the washing plant is mounted on its own separate pontoons (3). These are provided with winches, etc., for warping along the river bank as dredging progresses.

For certain operations there are definite advantages in having primary treatment plant on board the dredge and the main treatment plant separate, either on shore or on a nearby floating plant (4). The material is first scalped on the dredge by a screen, which generally reduces considerably the quantity of material that must be pumped to the treatment plant, located either on shore or floating pontoon.

Such an arrangement allows full advantage to be taken of the jet-venturi-type solids handling pump, the material discharged being fed into a 'boiling' box which distributes the material over a scalping screen. (A low-head screen is preferable so as to keep the centre of gravity of the dredge as low as possible.) The undersize from this screen is fed into a settling cone immediately beneath. The cone can then be used to weir off excess water together with any unwanted slimes, the solids settling to the base of the cone, from which it is drawn off and pumped to the plant by a solids handling pump. To prevent the suction orifice becoming choked with sand, agitation at the base of the cone by water jet may be necessary.

Power requirements and pipeline and pump maintenance costs for this type of installation are substantially less as in certain applications up to 70 per cent of the excavated material above a certain screen size can be rejected prior to pumping. High-efficiency solids handling pumps and, in certain circumstances, rubber-lined pumps can be used, and the increase in efficiency thus achieved can amount to 40–50 per cent, resulting in substantial saving in the cost of prime movers, fuel and maintenance.

A further factor that warrants consideration is the careful control on the density of the feed this arrangement affords, allowing the maximum concentration of solids to be pumped through the line by the addition or rejection of water from the cone.

In operations where the treatment plant is unavoidably a long distance from the dredge, making the cost and maintenance of long pipelines uneconomic, barges are used to take the dredged material to the treatment plants, this arrangement being common in gravel-pit operations.

General factors in design

Site conditions have an important bearing on this type of dredge construction. Where operations are to be undertaken in remote areas with difficult access, or where it may be necessary to dismantle the dredge to move to new areas (e.g. to avoid rapids), sectional pontoon construction is an advantage.

The design of winch installed on a dredge has a considerable effect on its handling and thereby on the production. In certain applications hand winches can be used successfully where free-running material of considerable depth is all that is to be excavated. However, where material that is standing may be encountered, or where relatively thin layers are to be cleaned up, it is of importance that powered winches are installed if production is to be maintained. Thin layers of material will also require the dredge to be moved quite rapidly along the deposit as the face advances.

Generally, winch requirements will be variable owing to the changing nature of the excavation, and to meet them hydraulically powered winches would appear to offer some advantages over those powered by electric motors. The varying rope speeds needed can be achieved quite simply and compactly with hydraulically powered winches.

Control of the dredge in relation to the whole operation must be carefully considered with a view to giving the dredgemaster the necessary facilities to maintain required production without undue fatigue—for instance a central position for controls and the necessary hydraulic gauges, with good all-round view of the working area.

Selection of suitable pumps and pump ancillaries is, in addition to general hydraulic requirements for a definite duty, also governed by the type of material to be dredged and its effect on the pump and equipment.

Characteristics of the deposit and effect on operation

To meet the variety of material to be handled, the range of equipment extends from the small unit such as that shown in Fig. 4 to the large cutter suction type of dredge described in Table I.

Broadly speaking, materials can be divided into two general categories when considering dredging operations—those which are susceptible to the action of high-pressure water jets to move them and those that require mechanical means for their disintegration and conditioning prior to pumping. Owing to the stresses involved in cutting the material a dredge equipped to deal with the latter category is substantially heavier than the straight suction dredge.

In addition to their use for breaking up material, rotating cutters have also been employed in relatively free-running material to break the inertia of the material so that it flows more readily into the suction pipe when the duty on the suction side of the pump is high owing to excessive depth. Recent developments in the use of jet-venturi-type pumps, however, overcome the limitation of centrifugal pumps with regard to depth of effective dredging, and have application where the material is suitable, but rotating cutters should always be fitted where there is any doubt as to the ability of hydraulic nozzles to disintegrate the material unless a substantial reduction in output can be accepted.

High-pressure water jets have, however, been used with considerable success for disintegrating quite hard materials in dredging operations and can often prove efficient where cementation occurs. In action the jet of water moving over the surface of the material finds a point of weakness and washes out a hole. In reasonably firm material the water pressure builds up within the hole formed until it is sufficient to overcome adhesion and the surrounding area is broken out along planes of weakness. Hydraulic jets are least effective in plastic clays where the action is restricted to washing away only the immediate area which is in contact with the water jet.

Hydraulic tests on marginal materials are the only sure way in testing this method of disintegration and should be carried out submerged owing to the reduction of effectiveness of the jets in water.

Output of the operation is influenced by the percentage of material too large to pass through the dredging system, and problems of blockages increase with the use of smaller dredges which have smaller suction pipes and reduced clearances in the pump. In many applications it is necessary to install a larger size of suction dredge to allow for the downtime resulting from stoppages from this cause. The amount of time lost increases with dredging depth.

Blockages due to the particle entering the suction orifice and jamming somewhere within the suction pipework or pump can be considerably reduced by fitting a reducing ring on the suction orifice which decreases the effective diameter at this point by, say, 1 in. Where only occasional large particles are expected which may cause a blockage this can be overcome by reducing the diameter at the suction orifice and by providing suitable clean-out points in the suction pipework and ahead of the pump. Similarly where there is a marked difference between the diameter of the suction pipe compared with the largest particle that the pump can pass, the amount of downtime can be reduced in installing a 'stonebox' ahead of the pump.

However, great care is necessary in assessing the effect of oversize in a deposit on the output achieved by a suction dredge. Various ancillary equipment, such as the travelling screen and to a certain extent the rotating cutter, are available for removing oversize from the suction orifice. Where considerable quantities of oversize are expected this must be removed rather than displaced and in order to prevent the formation of a layer of oversize material which would prevent the suction pipe from being lowered additional equipment is required. A similar problem arises when sunken trees and vegetation are encountered; where output is adversely affected, inspection by divers should be employed together with rock-grabbing equipment on the bows of the dredge.

Underwater investigations and bedrock clean-up

As well as evaluation for diamond content before the operation can begin it is essential that the material should be assessed as to size and the possibility of oversize.

A system of underwater pitting was developed in which airlift pumps were employed for excavating the material from within sectional casings, their control being effected by divers.

The equipment used for these investigations of the deposit prior to the installation of dredging equipment was also employed as a production unit in areas where operation of the dredge was impracticable.

The larger of two separate pontoons carried the heavy equipment, comprising compressor and plant for processing the excavated material, while the lighter pontoon served as the diving platform and provided equipment for manipulating the airlift used for excavating the material. Both pontoons were provided with A-frame and winches, on the smaller for manipulating the airlift pump and on the larger for positioning the sectional casing used for supporting the sides of the excavation.

The casings were first assembled on the smaller pontoon and then lifted and lowered to the bottom by the A-frame of the larger floating unit. The diving and airlift unit was then centred above the casing and the suction of the airlift lowered. A diver then operated within the casing manually controlling the airlift suction head and keeping contact with surface by telephone. Control of the airlift position was thus easily effected and in addition new casing could be lowered as necessary by personnel on the surface.

On these preliminary operations it was possible for the diver to see the type of material entering the suction of the airlift and the overburden was discharged over the casing side on the downstress side to waste. When payground was reached this was fed into hoppers for measuring and then supplied manually to the processing equipment on the second pontoon. With the use of small-diameter airlift the limitations of lift above water-level prevented a direct feed being made, but more convenient feed arrangements at shallow depths can be achieved using jet-venturi-type pumps.

The lack of visibility below 20 ft, in the tests undertaken, was a major problem, as at such a depth the visibility decreased rapidly due to the fine particles in the water until at depths of around 25–30 ft the visible distance was reduced to below 1 ft. Tests for increasing visibility undertaken with floodlighting equipment below 50 ft were unsuccessful as the particles blanketed the lighting effect and also reflected back in a manner similar to a thick fog.

In underwater operations in a deposit comprising a thick layer of free-running sand overlying a thin layer of gravel resting on decomposed bedrock, it was found necessary to drive the casings into the sand to prevent it from continually flowing into the excavation under the edge of the casing. To achieve this, the compressor employed for supplying air to the diver (and also air for the airlift) was used for powering a compressed-air jack-hammer which was periodically used on the rim of the casing to drive it into the sand. Similarly, when the first casing had been sunk to its full depth a second casing was entered in the first, the edge being hammered pneumatically as the material was excavated from inside.

Oversize material, which in those particular operations was considerable, was removed manually by the diver and loaded into a perforated bucket for hoisting to the surface.

Table III gives details of the equipment used in this undertaking.

TABLE III.—*Details of prototype underwater prospecting equipment*

Pontoons: one large unit carrying the compressor and ore-dressing equipment and one small unit used as diving platform and for manipulating airlift
Compressor: diesel-powered unit, delivery 100 ft^3/min at 100 lb/in^2
Diameter of airlift: 4 in.
Maximum depth of operation: 47 ft
Estimated output of airlift: 10–20 yd^3/h
Diving equipment: 2 'dry suits' with full face masks; two-way telephone; compressed-air filter arrangement from the compressor incorporating two emergency air bottles in case of compressor failure
Diameter of casings: maximum 6 ft 6 in.; minimum 5 ft

For the clean-up of mineral-bearing ground resting on hard irregular bedrock there would appear to be considerable scope for the application of similar equipment in exploiting such deposits in order to achieve the maximum recovery of minerals. Highly mechanized stripping and mining operations of dry opencast operations are often followed by slower but more selective and thorough hand-cleaning operations; but excavation of minerals from below water-level by dredges has not been completed by such an operation in the past owing to the difficulties of operating such a technique under water.

The problems involved in such an undertaking should not be underestimated, but considerable advances have been made in overcoming them, notably among diamond mining operations being undertaken off the coast of South West Africa.

In Far East bucket-dredging operations the problem of bedrock clean-up is aggravated by the large lower tumbler and sagging configuration of the bucket band which prevents the entry of the ladder into narrow fissures and between bedrock pinnacles.

Prospecting of such a potential dredging area by drilling or pitting normally shows up the general nature of the bedrock, and from this information a dredge factor is applied to the value of the area. This normally reduces the recoverable mineral—sometimes by a considerable amount. Examples are common where ground worked by a bucket dredge has been reopened and reworked by dry opencast methods, recovering the mineral left in the fissures and between pinnacles in the bedrock.

Recent operations, some of which have been described in this paper, indicate that in certain conditions the dredge factor, or amount of mineral that can be recovered, can be increased substantially by the use of such secondary clean-up equipment employing airlift and jet-venturi pumps controlled by divers.

REFERENCES

1. Mero, J. L. *The mining and processing of deep-sea manganese nodules.* (Berkeley: Institute of Marine Resources, University of California, 1959), 96 p.

2. More, G. A. Suction-cutter dredging for tin in Malaya. *Trans. Instn Min. Metall.*, **39**, 1929–30, 56–129.

3. Scott, N. S. The Caland dredging project. *Trans. Canad. Inst. Min. Metall.*, **64**, 1961, 478–87.

4. Hill, J. C. C. Pumping solids by jet venturi. *Mine & Quarry Engng*, **28**, July 1962, 304–11.

622.362.4(425.2)

Paper 4

Gravel Operations in the Trent Valley, Great Britain

R. WHITESIDE, B.Sc. (Tech.), A.M.I.C.E.
Technical Director
T. D. PATERSON, A.M.I.Mech.E., C.G.I.A., A.M.I.Plant E.
Group Production Manager
I. H. McLEAN, A.C.S.M., A.M.I.M.M.
Prospecting Engineer
Hoveringham Gravels, Ltd., Nottingham, Great Britain

SYNOPSIS

A brief history is given of the development and integration of a group of quarries in the Trent Valley, located between Nottingham in the southwest and about 6 miles downriver of Newark in the northeast.

Some geological information about the formation of the river gravels under discussion and of the type of deposit is given, with a description and analysis of the various types of rock to be found and the crushing strength and aggregate crushing values of typical samples.

Various methods of extraction and of transportation to the processing plants are compared, methods of processing at various levels of production are outlined and some comparison of the types of equipment used are made. Quarry costs are referred to.

Requirements of the planning authority concerning restoration are set out, and the partial and complete restoration of wet pits and use of pumped fly ash as a filling medium are described.

Operation of the transport fleet is briefly outlined, and there is a description of maintenance arrangements and procedures for both mobile plant and vehicles.

INTRODUCTION AND LOCATION

Before 1948 there were several small gravel companies operating quite independently in the section of the Trent Valley which forms the basis of this paper, some of them for many years. In 1948 the gravel quarry at Hoveringham, which by that time had been operating for about ten years, came into the ownership of what has ultimately become Hoveringham Gravels, Ltd., and was rapidly expanded and developed.

During the period from 1948 to the present time the Company has expanded its interest in this area by the purchase of other existing businesses and now controls eight gravel quarries in the area extending from Nottingham in the southwest to about 6 miles downriver from Newark in the northeast. The sites vary considerably in size—ranging from a plant capable of 200 tons/h to one of 40 tons/h. The main feature, however, of this particular group of quarries is the availability of large-scale reserves of high-quality gravel, very considerable production capacity and, through integration of management, extreme flexibility of operations in the production and delivery of material.

The area in which the quarries are situated is principally agricultural (Fig. 1). The city of Nottingham and the ancient town of Newark provide a local market, but the economics of road transportation and the quality of material produced mainly determine the extent of the area served by the quarries, which includes several adjacent counties. In the case of Girton, material is water-borne down the Trent to Hull and the Humber. Generally speaking, the major part of material produced is 'exported'.

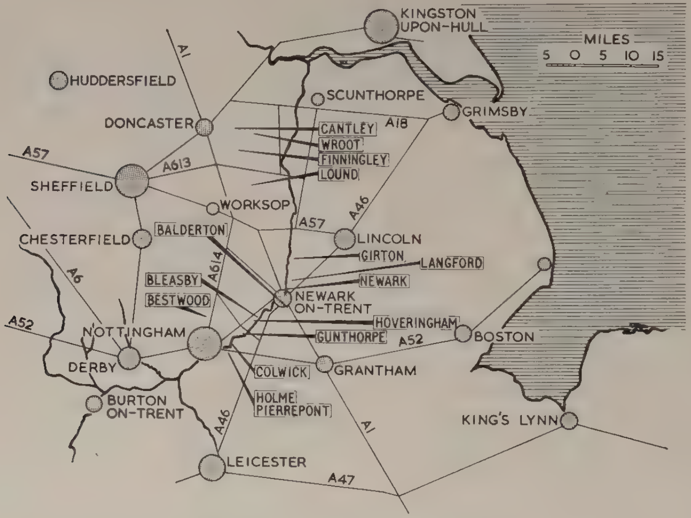


Fig. 1.—Trent Valley service area.

GEOLOGY

Geologically, all the deposits are Flood Plain Terrace gravels in the newer drift of Pleistocene age. Older Pleistocene gravels, Hilton and Beeston Terraces, are worked by others upstream of Nottingham. A Beeston Terrace deposit exists a few hundred feet south and above the Holme Pierrepont quarry, but this is not worked by Hoveringham. The deposits represent the last phase of the results of two periods of glaciation, the Pennine Drift and the Eastern Glaciation, with the subsequent alteration in the Trent River system caused by flows of waters derived from melting ice in inter- and postglacial periods, as shown in Table I.[1]

TABLE I

NEWER DRIFT	River Terraces	
	Flood Plain Terrace—Trent No. 1:	
	(*a*) below Beeston; 5–10 ft above alluvium	
	(*b*) above Beeston; below modern alluvium	
	Beeston Terrace	
OLDER DRIFT	Lower Hilton Terrace	
	Upper Hilton fluvioglacial terrace gravels	
	Outwash gravels of retreating Pennine Ice	
	Deposits of Eastern Glaciation	Chalky boulder clay and associated outwash gravels; Middle Pennine boulder clay
	[Long time interval]	
	Pennine Drift	Skegby outwash gravels; Early Pennine boulder clay

The gravel thickness varies from 9 to 30 ft under overburden, depths averaging about 6 ft. Average size of gravel in the ground is $\frac{1}{4}$ in. to 3 in., although larger sizes are present; these comprise less than 5 per cent. The proportion of sand ($-\frac{3}{16}$-in material) to gravel is roughly 50:50.

Approximately 80 per cent of the material has been derived from the erosion of Bunter Pebble Beds, mainly quartzite. In petrographic analyses[2] of typical samples, two types of quartzite occurred, only differentiated by their colours, one being pale and the other liver-coloured; otherwise, they were chemically and physically similar in respect of their use as concrete aggregate material. The pebbles were well rounded, of high crushing strength and generally less than $\frac{3}{4}$ in. in size. Vein quartz in the same samples was also found in the form of hard, well rounded pebbles from $\frac{1}{2}$ in. down to $\frac{3}{16}$ in. in size. Flint was found in the same size range, but was angular in form.

Erratics in the Valley include sub-angular pebbles of Carboniferous and Liassic limestone, chert and flint, Millstone Grit, Carboniferous and Keuper sandstones, granite and basic igneous rocks. The form of these erratics varies in shape from well rounded to angular showing some conchoidal fractures, hardnesses and textures also varying widely.

---

[1] Posnansky, M. The Pleistocene succession in the Middle Trent Basin. *Proc. Geologists Ass.*, **71**, 1960, pt. 3, 285–311.

[2] Evans, W. D. Department of Geology, Nottingham University. Private report.

Results of petrographic analyses (*b*) and of another analysis carried out on material from Holme Pierrepont (*a*) are given in Table II.

TABLE II

Weight %	*Sample A*	*Sample B*	*Coarse fraction of B (greater than $\frac{3}{8}$ in.)*	*Fine fraction of B ($\frac{3}{8}$ in. to $\frac{3}{16}$ in.)*
(*b*)				
Quartzite	58·2	50·2	46·9	56·8
Vein quartz	20·1	20·8	22·4	17·9
Sandstone	10·7	16·4	18·8	12·3
Flint	5·9	6·5	5·5	7·5
Miscellaneous	5·1	6·1	6·4	5·5
	100·0	100·0	100·0	100·0

(*a*)	%
Bunter Pebbles	82
Flint	13
Carboniferous and Derbyshire erratics	2
Jurassic erratics	Nil
Others	3
(No. of pebbles counted, 347)	100

Unlike the older gravels upstream the proportion of erratics from Derbyshire and the west is low. This means, in practice, that the greater part of the aggregate produced ranges in shape from well rounded to sub-angular, as shown by Figs. 2, 3, 4, 5 and 6, and is of high strength. Aggregate crushing values range from 13·5 to 16. The crushing strength of individual quartzite pebbles is about 63 000 lb/in².

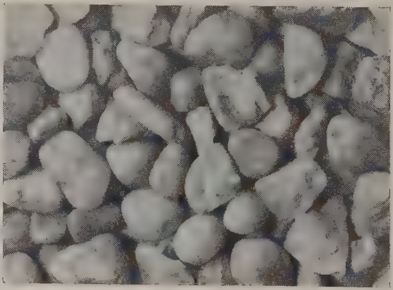


Fig. 2.—Vein quartz fraction: well rounded and only infrequently broken to reveal conchoidal or glassy fracture planes.

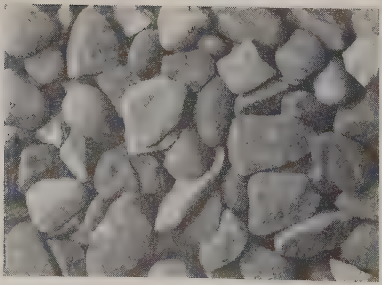


Fig. 3. (*left*)—Quartzite fraction: uniform size, but owing to the mixture of angular and sub-angular pebbles it produces excellent aggregation.

The shape characteristics described render the material eminently suitable for the production of concrete, giving a high degree of workability with the lowest possible water–cement ratio, which fact, taken with the high strength of the material, means that the aggregate produced is of a quality able to meet the most stringent specification required for structural works.

Investigation of gravel sites is being dealt with elsewhere, but often, owing to the comparative regularity of these deposits, it is not found too difficult to prove reserves, excavator pits sufficing in some cases. More detailed work involving percussion drilling and resistivity survey has, however, been done at Langford and Girton.

These Flood Plain deposits are comparatively regular in formation, although there can be marked differences in the size of material found in a particular locality, and the proportion of gravel to sand also varies.

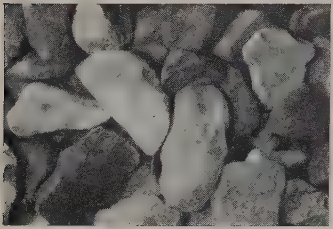


Fig. 4.—Flint fraction: typical shards and smooth pebbles of flint.

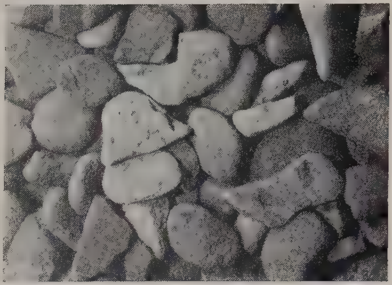


Fig. 5.—Sandstone fraction: well worn clean surfaces on the pebbles, together with a wide range of rounded and sub-angular shape-factors, causes them to aggregate efficiently.

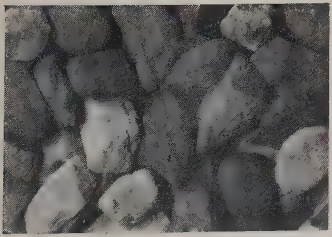


Fig. 6.—Miscellaneous fraction: the colour tones indicate the light-coloured granites and limestones; the darker rough-surfaced pebbles are of basic igneous rocks and metamorphic rocks, and the smooth black pebbles are of chert.

EXTRACTION AND PROCESSING

Removal of topsoil and overburden (Fig. 7)

Topsoil is normally removed by means of tractor and scraper and stored on some convenient area nearby for subsequent re-spreading during restoration. Where equipment used for this work is required to operate throughout the country, under varying conditions, the most suitable unit is the Caterpillar tractor and 9–11-yd^3 scraper, the horsepower being in the region of 125–160. Overburden can also be removed by this equipment when it lies above the water-level. Opinions differ as to how deep one should go with tractors and scrapers, but normally 6 ft would be the limit, after which excavators would be employed.

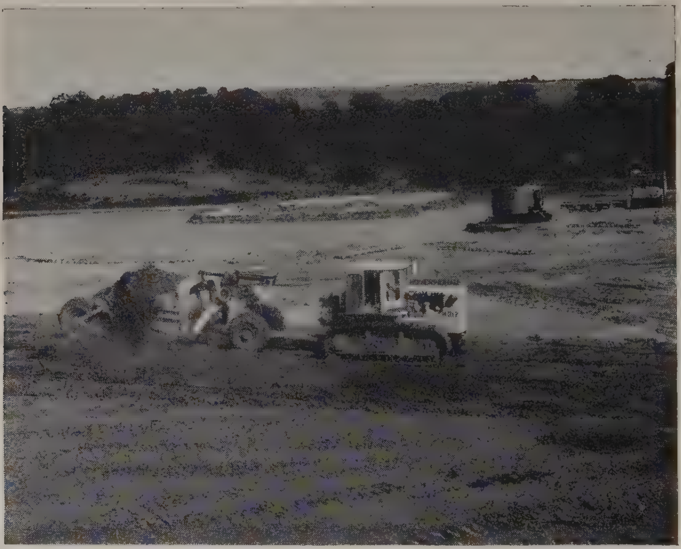


Fig. 7.—D 7E Caterpillar tractor and scraper removing overburden at the gravel face near the River Trent.

Muck removed by excavator is cast back into the worked-out face, the machine size being sufficient to ensure that it is thrown well clear of the line being formed by the new face. This system works well when circumstances permit and has the obvious advantage that the muck is only handled once.

The cost of earthmoving by tractor and scraper varies according to the material and the distance it has to be hauled.

Method of extraction

In the Trent Valley the raw material is below the normal water-level and extraction is by excavators fitted with dragline buckets or, in one case, by a gravel pump.

The excavator is by far the most suitable and positive method. The size of machine varies between $\frac{3}{4}$ yd^3 and $2\frac{1}{4}$ yd^3, depending on the method of dealing with overburden, the depth of the material and the outputs required (Fig. 8).

The gravel pump, though efficient under ideal conditions, leaves much to be desired. The cost of replacing wearing parts can be high and the limit on the distance the material can be pumped requires the use of a second pump or some alternative method, such as conveyors, to get the material to the plant. Given a completely free-running material an 8-in pump can deliver up to 300 yd, but such conditions are seldom met and delivery lines are usually much shorter. In addition, blockages can occur which result in considerable delay and loss of production.

Having decided that the best method of extraction is by means of the dragline, the handling of wet material presents problems. Access from face to plant must be kept fairly dry and material reaching the plant must have a limited moisture content to permit it to travel on the feed conveyor. To satisfy these requirements it is common practice to cast the material from the face on one side, leaving it to drain off, then reload it by means of the same machine or a second one if production

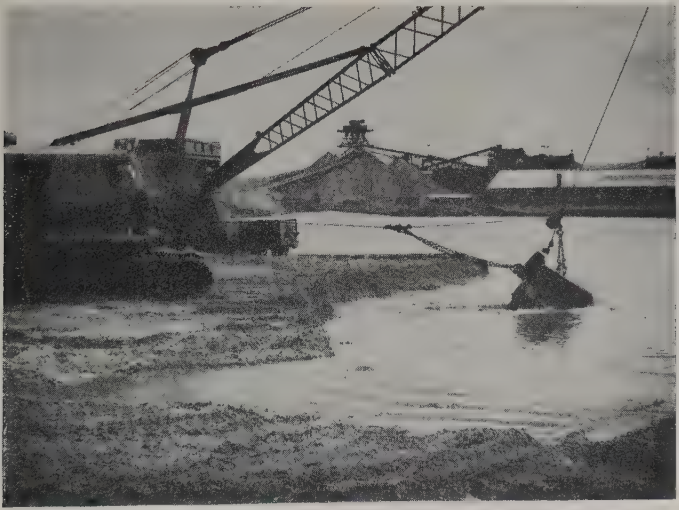


Fig. 8.—38 RB excavator with dragline working from a wet face at Hoveringham quarry.

requirements demand it. Excavator buckets can be drilled or cut to assist drainage during extraction from below the water-level, but only to a limited extent, as saleable sand and stone of small size tend to be lost by the washing action as the bucket is drawn through the water. Some benefit can be obtained from operating below the water-level if the material is particularly dirty, as it is given an initial wash, but, generally speaking, this is not necessary in the Trent Valley and the disadvantages far outweigh the advantages.

Perhaps the main point against wet working is the fear that the excavator driver, however efficient he may be, is leaving good-quality material behind in a face which cannot be seen. There must be considerable difficulty in 'bottoming up' and at the same time avoiding getting clay in the gravel. Faults too are difficult to work round when nothing can be seen, and time is lost in dragging the excavator bucket out only to cast useless material back, until probing around again finds the gravel.

It is quite common practice to pump faces dry when the total depth is in the region of 10 ft. In the Trent Valley, where depths of 20 ft or more have to be contended with, the problem is greater, but recent

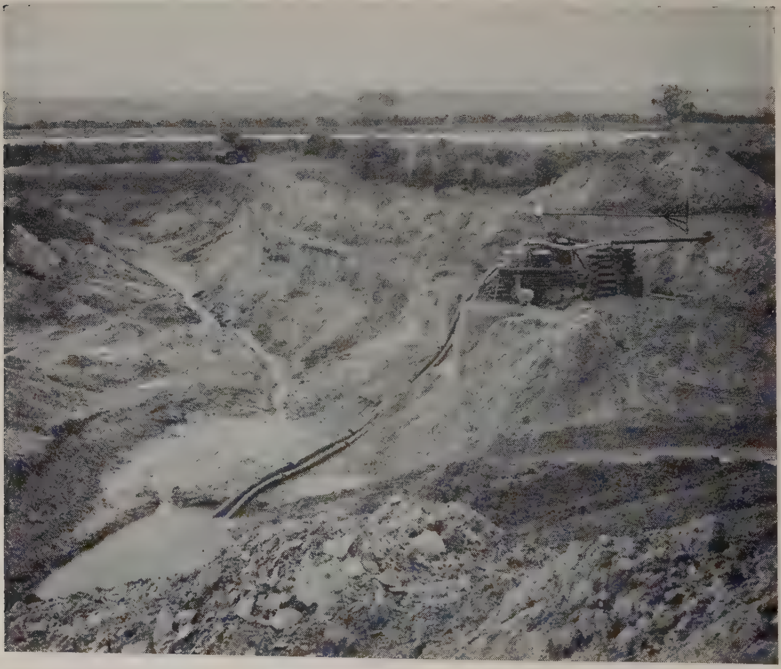


Fig. 9.—Sykes Univac 6-in pump operating on the dry face at Hoveringham quarry.

advances in water-pump design have now made dry working in several quarries possible by enabling the face to be sealed off in areas of reasonable size. The benefit was immediately felt as the use of excavators during the night to cast out wet material for draining could be discontinued, and the clay bed on the quarry floor is now clearly seen as the excavator bottoms up. Access roads remain dry and the material is quite suitable for immediate handling by quarry transport or plant conveyors (Fig. 9).

Transport of raw ballast from face to plant is by several methods. The locomotive and tub system still in use, each loco hauling five or six 2-yd^3 tubs, has the advantages that it allows the material taken from the wet faces to drain on the way to the plant and that weather conditions do not interfere with progress to the same extent as with other methods. However, the initial cost of locos, tubs, rails, etc., is high, the labour necessary to maintain the track and equipment in good order is costly, and considerable work is involved in moving from point to point as the excavator moves along the face. In general, it is not a flexible system and is losing favour in the Trent Valley.

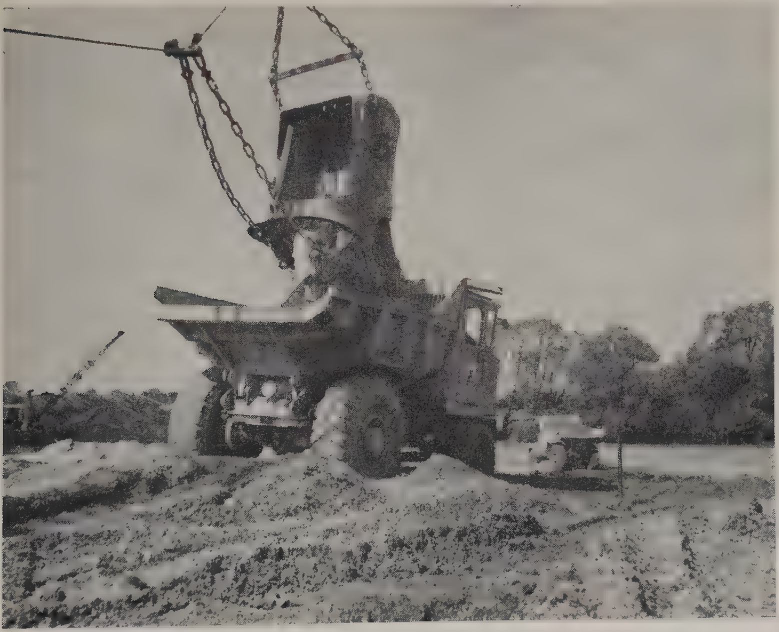


Fig. 10.—Aveling Barford SL dumper, capacity 13½ tons, being loaded at Newark quarry.

Some 200-ton barges and tugs are also employed in conveying the ballast from face to plant, a useful method over large areas of water. Though positive enough, it is slow and the material has to be double-handled, being loaded and unloaded at both ends by machine. The cost of plant and labour therefore tends to be high.

For the larger operator who moves mobile plant from quarry to quarry at intervals to suit production demands, the dumper comes into its own (Fig. 10). This flexibility extends also to its use on sites for removing overburden, feeding raw ballast to the plant receiving hopper or stacking 'as raised' or processed material as required. Ground conditions can affect the dumper considerably, but a lot can be done to help by putting in a well constructed hardcore or hoggin main road and by carefully choosing the type of machine best suited to particular conditions.

Belt conveyors are also in use and, where distances are considerable and ground conditions bad, they are very efficient. New light-weight sections, easily coupled together, reduce the time of re-positioning these conveyors to a minimum and very little labour is involved in their operation.

Processing the material (Fig. 11)

Quite a range of equipment is involved in the Trent Valley in processing the raw material. Certain refinements have been added to various plants to suit particular requirements but, generally speaking, the types in question can be divided into two categories: those producing single-sized materials and the smaller, simpler unit producing 'down-sized' materials.

Dealing with the larger, high-output plants which produce single-sized materials to be batched to any particular B.S.S. specification, the feed hopper into which the ballast is tipped is constructed of steel and fitted with a scalping grid, the aperture of which is usually 6–8 in. square. From here the material passes to a short feeder which in turn loads the main belt to the washer barrel. It is at this point that the sand is separated from the stone by means of a $\frac{3}{16}$-in mesh screen—the sand being pumped, with the necessary water to hold it in suspension, to sand cones, each cone or classifier producing a particular grade of sand. Also at this point stone above $2\frac{1}{2}$ in. or 3 in. in diameter is removed by a screen and passed to a primary crusher where it is reduced in size and returned in closed circuit to the plant.

Material below 3 in. in size passes from the washer barrel to vibrating screens mounted over storage bins, suitably divided to take individual sizes. On the screens water jets are fitted to give the stone a second wash. On several plants two sets of screens are used, one to handle natural stone and the other crushed stone. The sizes generally produced are $\frac{1}{4}$ in., $\frac{3}{8}$ in., $\frac{1}{2}$ in., $\frac{3}{4}$ in. and $1\frac{1}{2}$ in. Any material above saleable size, i.e. $+1\frac{1}{2}$ in. to 3 in., is rejected from the vibrating screens and fed to a secondary crusher or crushers where the crushed stone is again returned to the plant in closed circuit. Thus crushing is carried out in two stages, the primary handling +3-in stone and the secondary −3-in to $+1\frac{1}{2}$-in. Fitted under each bin section are adjustable doors through which the various sizes of stone pass on to a batching conveyor and hence to a rinsing screen which directly loads into either road vehicles or further storage bins, each size being controlled by a travelling tripper in the case of the latter.

The type of crusher employed with these plants varies. Trent Valley material, being hard, is severe on wearing parts and crushing can be costly. Higher outputs and less wear are experienced with the cone-type crusher, although the material tends to be flaky in shape when passed through such machines, and better results can be obtained by choke feeding. A better particle shape is obtained with the impact breaker, but the cost of replacing wearing parts is greater. The jaw crusher is low in output and does not give such good particle shape as the impact breaker, but is useful for primary crushing when feed sizes are larger.

The smaller, less complicated, plants usually produce one or two

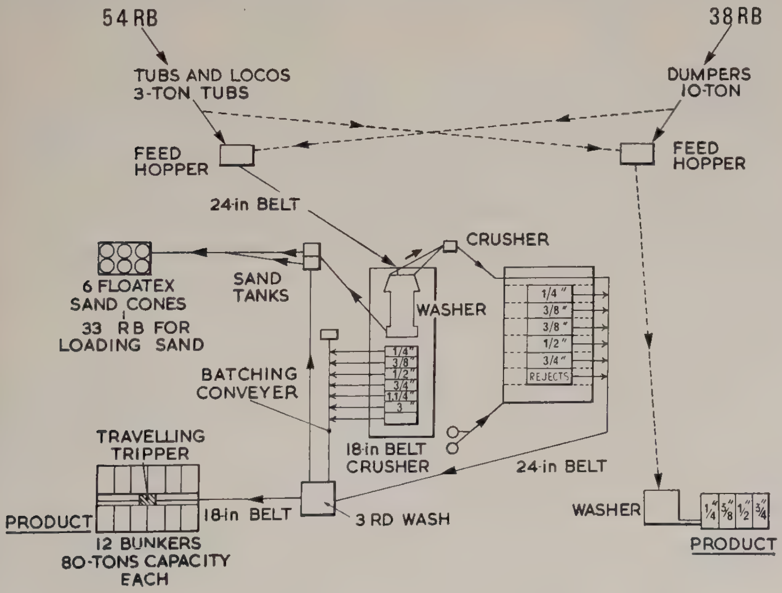


Fig. 11.—Typical Trent Valley flowsheet.

grades of sand, a $\frac{3}{4}$-in to $\frac{3}{16}$-in down-sized stone, $1\frac{1}{2}$-in to $\frac{3}{4}$-in stone and rejects ($+1\frac{1}{2}$-in material).

The sand is separated from the stone at the washer and usually passes to a sand wheel, as opposed to cones, for grading. The stone in the washer passes through to thimble or rotary screens fitted on to the end of the washer and is graded in the sizes previously mentioned.

This method has the disadvantage of having to wash and screen in one operation, but on certain quarries dealing with Trent Valley material quite satisfactory results are obtained at a level of production in the region of 40 t/h. Such plants are useful as boosters when demand temporarily increases. They are capable of producing single sizes, but their use in the Trent Valley is restricted to making 'down-sizes' —more straightforward and cheaper to operate.

Stocking

In the larger quarries bins are provided for stocking gravel, but invariably it is also necessary to allow for stockpiling on the ground. Sand is, almost without exception, left to drain on the ground as it falls from the towers of the classifying plant. Two or more heaps of each grade can be formed, which leaves at least one in reasonably dry condition.

Concrete areas are therefore provided for the purpose of stacking in addition to whatever bin space exists, so that the materials can be kept clean and loading plant has a reasonable surface to run on.

Loading

Materials are loaded into lorries direct from bins or by means of loading screens. The latter are very efficient and provide for water sprays, giving the material a final wash before being loaded.

Ground stocks are loaded either by mechanical excavator, fitted with grab bucket, or by rubber-tyred loading shovels. The excavator has the advantage of being able to work on high stockpiles and to mix materials to a specification more easily; the loading shovel is more mobile and in many ways faster.

Costs

Among the several factors affecting quarry costs is the economical use of plant. It is costly to have more plant than is absolutely necessary, or to have equipment which is in such condition, either mechanically or in design, that it cannot give adequate and reliable service, and this in turn will affect labour and plant hire costs on site.

Owing to increased demand in the industry generally, the nature of certain deposits and the planning conditions imposed on them, more earthmoving is taking place today than hitherto. The initial cost of the equipment involved, and the subsequent expense incurred in repairs and maintenance, call for tight control, both for economical operation and to ensure continuity at the face. Planned maintenance to quarry plant will reduce 'down-time' and also the cost of materials. Personnel should work to a routine which is regular and simple, irrespective of how hard-pressed the machines are.

As a guide, the following costs are given as a percentage of the total production cost:

	%		%
Direct labour	22	Depreciation of fixed plant and buildings	3
Mobile plant	29	Earthmoving (stripping and reclamation)	2
Maintenance labour	4	Land depletion/Royalties Rates General	26
Maintenance materials	9		
Light, heat and power	5		

RESTORATION

The extent to which restoration is carried out on any site depends to a large extent on the planning conditions as set out by the planning authority when a site is approved for development.

In the Trent Valley area these conditions vary depending on the date when the approval was given. Earlier development was permitted without conditions for full restoration, all that was required by the planning authority being that—

(1) topsoil and subsoil be removed and stocked on site for re-use in forming or covering the batter to the banks of any lagoons;

(2) all islands left in the excavations to be levelled off as directed;

(3) all necessary action to be taken to prevent nuisance and damage by growth of weeds whether within the workings or on stocks of soil;

(4) land to remain in agricultural use until actually required for mineral workings;

(5) so far as practicable, existing trees and hedges to be preserved;

(6) rivers, streams and natural drainage channels to be reinstated to appropriate levels to ensure their satisfactory functioning;

(7) the winning of surface minerals not to proceed within a specified distance of existing public highways, bridle or accommodation roads and the boundaries of the application; and

(8) before vacating the site, notice to be given to the planning authority, the whole site to be left clean and tidy, and all plant, machinery and buildings removed unless otherwise agreed that certain buildings could remain.

In later approvals permitting development the planning authority added a further condition that the land should be refilled to its original level and the topsoil spread so that it could be returned ultimately to agricultural use.

All the pits in the area under discussion are wet pits, which means that in normal circumstances, the natural groundwater level stands 4 ft 6 in. below the original ground level and a lagoon of considerable depth is formed. There are several disused areas, now lakes of 30–60 acres. The main difficulties in restoring land under these conditions are: first, to find material in sufficient quantity to carry out the work and, secondly, to ensure that there will be no deleterious effect from depositing the material in the lagoons.

The four principal sources of restoration material in the Trent Valley area are (*a*) local authority refuse, (*b*) colliery refuse, (*c*) Central Electricity Generating Board fly ash, and (*d*) sugar beet waste.

Of these, local authority refuse, which principally comprises household material, is not particularly suitable unless it can be laid in layers and covered with about 12 in. of inert material. It has its uses in the restoration of dry workings, but in the wet workings of the Trent Valley it can only be used to a very limited extent.

Colliery waste, on the other hand, is an inert material which would prove to be a very useful source of filling medium, but unfortunately it is not available either in the quantities or in the localities where it could be of the greatest use. The cost of transporting colliery waste to the particular worked-out areas would prove prohibitive.

Leaving aside sugar beet waste, the main source of fill available for restoration purposes in the Trent Valley at the moment is the C.E.G.B. fly ash. This has the same disadvantage as the colliery waste in that it has to be transported, but as the C.E.G.B. have no alternative sources for dumping they have shown great interest in filling certain of the worked-out areas in the Trent Valley.

The principal quarry involved in accepting fly ash is Hoveringham, where the C.E.G.B. have laid a pipeline from the Staythorpe Power Station for some 5 or 6 miles, the fly ash being pumped into worked-out areas at the present time. The first 20 acres or so of excavated area were filled by the beginning of 1964.

As the ash is pumped into the lagoons it settles on the bottom of the worked-out areas and the water, which is displaced, is led by ditches and dykes into the River Trent, great care being taken that no ash pollution occurs. Eventually, when a sufficient quantity of ash has been deposited in the lagoons the pipeline is moved on to an adjacent unrestored area and the pumping continues. The lagoon which has just been filled is left until it is sufficiently dry to take the weight of machinery when the topsoil is spread back on to the surface. At the moment the planning requirements are that 12 in. of topsoil shall be spread over the surface of the fly ash, although the Ministry of Agriculture claims that as little as 4 in. will be sufficient and have even conducted experiments to prove that fly ash is a suitable medium for growing crops without any actual cover at all. One of the difficulties that arises when the ash dries out is that in the event of any strong winds there is liable to be considerable dust nuisance, and for that reason the ash should be covered with soil as soon as possible.

Owing to the expanse of the various worked-out areas in the Trent Valley there is not at the moment sufficient ash to restore all the lagoon areas, but as the power stations increase in number it is hoped suitable arrangements can be made with the C.E.G.B. to introduce fly ash into the quarries at Holme Pierrepont and at Gunthorpe. In both these cases, though, the ash will have to be transported by lorry and tipped into the existing worked areas.

At the Newark quarry lagoons have been used for a number of years for the British Sugar Corporation's sugar beet factory adjacent to the quarry site. In the season large quantities of sugar beet washings have to be removed from the factory. Although the washings consist of soil with a high content of fertilizer, they are apparently not in great demand and are therefore discharged into worked-out areas on the quarry. On this particular site the gravel workings are in the flood plain of the Trent and the lagoons are also connected to the river by a cut used in earlier days for barging material. There is therefore a risk of serious river pollution from the fertilizers mixed in the sugar beet washings and solids in suspension in the effluent. In order to satisfy the River Board the lagoons receiving beet washings have been 'bunded' to stop river water from getting into these lagoons during flood conditions, and a fairly elaborate system for settlement within the lagoons has been provided to stop suspended solids from the beet washings being discharged into the river.

It may be of general interest to say that when the extraction process is completed the large stretches of water which remain provide a useful amenity suitable for fishing, yachting and water sports. Over recent years the demand for such facilities has considerably increased, all the open stretches of water in the area being used by angling societies and one also by a yacht club.

TRANSPORT

All transport in Hoveringham Gravels, Ltd., is controlled from group headquarters. The country is divided into two regions, each consisting of several areas. About 180 vehicles operate in the Trent Valley and trucks are allocated as need dictates, i.e. according to orders currently received. Wherever possible, the largest type of vehicle is used on the grounds of economy.

A problem with transport in the sand and gravel industry is that of keeping the vehicles fully employed. The number of vehicles required fluctuates from day to day and there is a seasonal decline in winter. In order to meet these requirements and to preserve maximum flexibility the company leases vehicles on an 18-month turnover period, which permits the strength of the fleet to vary by controlling the intake of new and the return of old vehicles. This arrangement provides for the special needs of the sand and gravel industry and is applied in practice to about half the vehicles in operation.

Before this scheme was adopted all company vehicles were purchased outright. The present arrangement has the advantages of—

(1) reducing the capital investment in road vehicles;
(2) meeting the seasonal variation in transport requirements;
(3) reducing major repair work on vehicles and thus breakdown time;
(4) providing greater flexibility in operation; and
(5) removing the problem of disposing of second-hand vehicles.

MAINTENANCE

The Trent Valley quarries, along with several others, are serviced by a large, well equipped depot where all repairs of a major nature are undertaken. Plant and transport in need of overhaul is replaced on site and the faulty unit brought into workshops for the necessary attention. Three low-loaders are available for this and for the general movement of equipment from quarry to quarry.

Mechanical work by quarry personnel is therefore restricted to minor repairs and normal daily maintenance.

Two modern high-pressure grease bays are available in the depot for day and night greasing of vehicles, but a small mobile unit, also fitted with pressure greasers, supplements these and enables certain vehicles to be greased on site during the busy season. This gives a quicker turn-round and avoids dead mileage in travelling to and from the depot. Items of mobile plant are serviced on site by operators in the normal manner.

All lubricants are obtained from one supplier and extensive surveys have been carried out to reduce to a minimum the number of different grades in use. This simplifies maintenance generally and helps to eliminate the possibility of error. Each machine and vehicle is supplied with a service chart showing the correct grade of lubricant, where it is to be applied and how often attention is required. Provision is also made for a record of what has been done.

Regular inspections are carried out on all equipment by a competent inspector who forms a link between quarry and depot.

Each vehicle undergoes a thorough external check of all components by depot personnel every 30 000 miles. At 150 000 miles vehicles are subjected to a further special check, involving more detailed work.

Plant and transport repair shops are under the control of an experienced works superintendent and foreman, as are stores, machine shops, welding shops, body repair shops, paint shops, etc.

Tyres are controlled by a tyre fitter who is provided with a workshop and specially designed van for emergency repairs. Strict records are kept of all tyre movements and costs, and regular inspections are carried out both by company staff and those of the tyre supplier. Reports follow these inspections so that whatever attention is necessary can be given in an orderly manner.

Vans, trucks and mobile welding vehicles are available in the depot to give quick service on site in the event of an emergency or impending breakdown.

Acknowledgement.—The photographs (Figs. 2–6) of the aggregate fractions are published by courtesy of Professor W. David Evans.[2]

Discussion

(Papers 1–4)

Mr. P. R. Harris,* introducing Papers 1 to 4, said that they covered all aspects of the sand and gravel industry, from finding and proving deposits in the first place, to the distribution of the finished product to the customer; but because they dealt with quite separate aspects of the industry he proposed to refer to the papers individually.

In Paper 1, 'Location and evaluation of sand and gravel deposits by geophysical methods and drilling', Mr. Vann described first the different types of equipment used in finding and proving sand and gravel deposits. The use of geophysics for that purpose was comparatively new to sand and gravel operations in the United Kingdom; it had helped not only to keep the costs of operations down but had very often given a more accurate evaluation of the deposit and its overlying overburden.

Two methods—electrical resistivity and seismic—were described and a guide as to which should be used in different circumstances was given. The author went on to describe different drilling methods employed, stating quite categorically that percussion drilling was still the best way of proving a sand and gravel deposit, but suggesting that a combination of geophysical readings and percussion drilling would give results accurate to ± 10 per cent at a much lower cost. Fig. 3 (p. 6) showed how the accuracy was achieved, together with the relative costs.

On the important question of costs, the paper gave comparative figures and the cost of electrical resistivity was approximately 25s. for each 50 ft of depth, with seismic methods 35s. for each 100 ft of depth (which, allowing for time lost in moving from each station, might average out at, say, 20s. for each 50 ft of depth), and for percussion drilling somewhere in the region of £75 for 50 ft of drilling depth.

From those bare figures the use of geophysics combined with drilling must obviously be the most economical method of proving the volume of the sand and gravel in the deposits as well as the volume of the overburden. The importance of the correct evaluation of the geophysical readings must, however, be stressed. It was a very simple matter to train a person to take the readings in the field, but the correct interpretation of those readings was something in which experience and comparison played a very important part, and it was important to use only those people who had the knowledge and the information to help them in their interpretation.

---

*A. Braithwaite and Co., Ltd., Knaresborough, Yorkshire.

Finally, the presentation of the results of a survey were fully described in the paper, with the use of thickness contours on plans given as a method of showing the client in a very simple and straightforward way the depth of the sand and gravel and overburden over the site.

In 'Use of modern floating grab dredges for sand and gravel extraction in the Upper Rhine Valley' Dr. Konz gave the reasons for its evolution—reasons for the great depth of the deposits (up to 300 ft), the restricted amount of land available and the consequent increase in prices for that land, and the 400 per cent increase in demand in the 10 years 1950–1960.

The first type of floating grab dredge developed was that with the simple luffing jib which worked through a well in the pontoon structure and tilted forward to discharge into a bin for feeding on to floating conveyors or a chute for loading into barges. That type of dredge had the disadvantage in that in shallow deposits the restricted production area meant that the dredge and its ancillary equipment had to be moved very frequently. That, together with the development of, and need for, grabs with the larger capacity, brought the overhead-gantry type of dredge into being.

The special winch system was described in detail—its automatic control over the closing of the grab, the depth to be dredged and the height the grab was raised resulting in ease of operation and strict control over the production operation.

Hand in hand with the development of larger and more automated dredges, came the development of the floating conveyor and the author explained the difficulties in the use of those units and described the operation of the bottom-dump barges with dredges of different types and the methods of reclaiming that material after the barges had deposited their load.

Brief reference was made to dredges incorporating the complete processing plant, with its consequent advantages and disadvantages. The speaker also wished to draw particular attention to Tables I and II (pp. 25 and 37) giving operating details of grabs of different capacity working on different deposits and a breakdown of the production costs, respectively.

Mr. Hill did not refer specifically to sand and gravel operations in his paper 'Some developments in connection with the use of suction dredges for mining alluvial deposits', but did say that the experience gained in diamond mining operations could be applied to general mineral dredging. Details of the use of that equipment—from overburden stripping operations in Canada, where a 400-ft depth of material was removed from a deposit of iron ore, to smaller units designed for the extraction of mineral-bearing gravel from river beds—were given, and it was the latter type of equipment which was of interest to sand and gravel operators in the United Kingdom.

The paper referred to specific types of deposits where suction dredging was more successful than other methods and, in particular, to offshore dredging, where it had the advantage of being able to dredge to considerable depths. The smaller unit, described on page 45, could, for instance, with the addition of pontoon and pipe sections, dredge to a depth of 100 ft below the water-level.

Details were also given of suction dredging equipment used for winning mineral-bearing deposits incorporating a jet-venturi-type pump for elevating the material from the river bed, and a centrifugal gravel unit for delivering the mineral-bearing gravel to the treatment plant on shore. In that way the dredge might be used to maximum efficiency for its primary purpose of deep dredging.

That particular equipment was described fully in its operation on fast-flowing rivers in West Africa. The problems and hazards of the operation and the methods of overcoming them are explained, together with the surveys carried out to determine the type of material that would be found.

The author also discussed the design of suction dredges, giving reasons for the different designs which were suited to different types of operations and different deposits, and the equipment and methods used to determine the size of material in a particular deposit.

In ‘Gravel operations in the Trent Valley, Great Britain’ the authors gave, first of all, a brief history of the development and integration of the Hoveringham Gravels, Ltd., group of quarries.

The deposits varied in depth from 9 to 30 ft, with an average overburden of 6 ft. The maximum size of gravel was, in the main, 3 in.

The proving of the deposits had been comparatively simple, owing to their uniformity—in many cases pits dug by excavators had been sufficient—but in certain areas more detailed investigations had been carried out using the combination of percussion drilling and electrical resistivity as described by Mr. Vann. Topsoil in the deposits was removed by tractor and scraper and stored for subsequent restoration, while the subsoil was removed by dragline and cast back into the worked-out part of the pit clear of the new face.

The paper compared the use of gravel pumps and dragline excavators for excavating the sand and gravel and, because of the high cost of repairs and restrictions on the distance it could pump the material, the gravel pump lost favour to the dragline, which was used for all gravel winning. The problems of underwater dragline operations were discussed, and the use of highly efficient pumps to lower the water-table was described.

Practically every kind of transport from the face to the processing plant was used in the area and the advantages and disadvantages of the different methods were dealt with. A typical processing plant, together with the types of equipment used, was also described.

Details of restoration difficulties were given, together with a description of the organization of the transport fleet and its maintenance.

Mr. P. W. Dobson* said he would first like to thank Dr. Konz for his very interesting and well illustrated paper.

Having regard to the general trend towards higher hourly outputs in modern dredging equipment using grabs of 6- or 8-yd^3 capacity, did Dr. Konz expect that higher hoisting speeds would be employed in the future? It appeared that current German practice was to hoist at speeds up to approximately 250 ft/min, which did not seem particularly ambitious, although it was accepted that there were many factors to be taken into account in selecting hoisting speed.

Work in the United Kingdom suggested that trouble might be encountered due to 'washout' of material at very high speeds, but the indications were that speeds of 400 or 500 ft/min would be quite safe from that point of view.

If higher speeds were to be employed in the future, did the author agree that complete automation of the cycle would be essential to enable the shorter cycle times to be maintained consistently throughout the workshift? Was there any activity towards that end in Germany? The author had touched on the subject of automation, but perhaps that could be pursued a little further.

Mr. R. Whiteside† said he was very interested in Paper 2 by Dr. Konz, particularly as his own Company had extensive workings of sand and gravel in the Trent Valley where, no doubt, many of the problems faced were similar to those in the Rhine.

He wondered what Dr. Konz would consider to be the minimum working depth at which a grab dredge could operate. He appreciated that the deposits in the Rhine Valley were extremely deep by British standards (in some cases being well over 200 ft, whereas in the Trent Valley the maximum depths were no greater than about 30 to 35 ft), and he wondered whether, in view of the much shallower deposits found in the Trent Valley, the author considered that a grab dredge would be an economic proposition.

---

*Head Wrightson, Ltd., Yorkshire.

†Hoveringham Gravels, Ltd., Nottingham.

In the Trent Valley several large worked-out areas were being reclaimed by using waste ash from power stations, that material being pumped from the power station along a pipeline to the site of the worked-out areas, where it was discharged into the large open stretches of water. Was anything comparable to that done in the Rhine Valley and to what extent were the workings along the Rhine controlled by legislation insofar as restoration was concerned? If the areas excavated did not have to be reclaimed, he would be interested to know whether they were used for any amenity purposes, such as fishing or yachting.

In the United Kingdom land was often owned by the gravel operator, although in many instances it was worked on a royalty, and he would like to know who owned the land being worked in the Rhine Valley. Perhaps the author could also describe the common practice in the Rhine Valley.

He understood from the paper that the material, having been dredged, was transported to a shore station where it was processed before often being reloaded into larger barges for delivery to markets lower down the river. What capacity of barge was used for that purpose; was the barge affected by locks; and were there any special river dues? He would also like to know whether Dr. Konz had considered any special methods of off-loading barges at their destination. That was commonly carried out by using an excavator fitted with grab bucket, but in his opinion could possibly be carried out more quickly by using a conveyor system specially fitted into the bottom of the barge or by drag scraper, and he wondered if Dr. Konz had had any experience of those methods.

Mr. I. H. McLean,* referring to Paper 1 by Mr. Vann, said that because the only absolutely accurate method of assessing any deposit was by working it out in its entirety, and as resistivity applied to gravel assessment was a relatively recent technique, he wondered how the computations of error as reported were arrived at.

In Bunter Pebble Bed assessment, having established by physical means the presence of stones at a certain horizon, would it be possible to follow that horizon by a constant separation electrode method? The shell and auger drilling cost quoted by Mr. Vann (20–30s. per foot) was presumably the net rate per foot charged by a contractor to a client; that would be grossed up to more than £2 per foot by the inclusion of on-site and erection and dismantling charges. His own Company's cost of similar drilling averaged at 26s. per foot over the past twelve months and the author's comments would be welcomed. He would also mention that his Company's average overall prospecting and assessment cost over the past 2½ years was of the order of £6 per acre.

---

*Hoveringham Gravels, Ltd., Nottingham.

Finally, he would be interested to learn if any work had been done on the correlation of sample results from down-the-hole air flush drilling and a face sampled control, in regard particularly to the relationship of particle size analysis of the air flush samples to actual fact, as established by pitting or channel sampling a face.

Mr. J. A'C. Bergne,* referring to the evaluation techniques described in Papers 1, 5 and 7, pointed to the elaborate published details of sampling procedure in alluvial gold and tin mining and to the paucity of similar information for sand and gravel evaluation. It seemed that once an auger or other tool had penetrated the deposit, a veil was drawn over the rest of the procedure.

It was possible to describe sampling methods and calculations for sand and gravel without betraying the relative values of deposits and he hoped the authors would feel able to elaborate on that aspect of their work, though admittedly his question was a leading one.

Mr. N. Brindle† asked if Mr. Vann could explain how the percentage accuracy referred to in his description of resistivity methods of prospecting was determined. If great accuracy in volume determination could be guaranteed by sufficient work, was that really of value if the quality of the deposit was not accurately known?

During the past few years his own Company had completely worked out several gravel deposits and it had been possible to compare estimates of reserves made previously by electrical resistivity methods with the tonnages actually realized. Those 'acid' tests had shown that although the volumes of the deposits were usually accurately determined, quality variations (excessive clay silt or lignite) frequently rendered parts of the deposits worthless. Present practice in his Company, with any terrace gravel deposit less than 20 ft in depth, was therefore to drill using a single-flight auger. The costs of that method were similar to those of a resistivity survey and, with care, a good indication of quality could be obtained.

---

*Independent Consultant.

†The Associated Portland Cement Manufacturers, Ltd., Stone Castle, Greenhithe, Kent.

With regard to the Bunter deposits, in his experience the principal gravel beds were fairly continuous deposits and, by combining geological mapping and drilling, certain of the gravel beds had been traced for several miles. Careful geological mapping, supported by drilling to confirm the structure and depth, had now been adopted as the basis for the determination of gravel reserves.

Mr. W. Potter,* referring to Paper 1 by Mr. Vann, said that the author stated that the seismic method would be more accurate if hard rock underlay the gravel, since lack of contrast would make electrical resistivity inaccurate. Could the author suggest any other factors which would diminish the contrast of resistivity surveys?

He would also like to know if resistivity surveys located the water-table, if any, and if core drilling had any application in such investigations.

Mr. R. O. Warburg† welcomed Mr. Vann's paper, but expressed disappointment that the author had not drawn attention to the need for a geological or geomorphological appreciation of individual sites before any programme of prospecting was initiated or to its necessity in some cases in deciding the location of confirmatory bore-holes and in interpreting the results of a geographical survey.

To take an example, a site in the Thames Valley might consist of gravels at two terrace levels—Boyn Hill and Taplow—with an intervening bluff of London Clay.

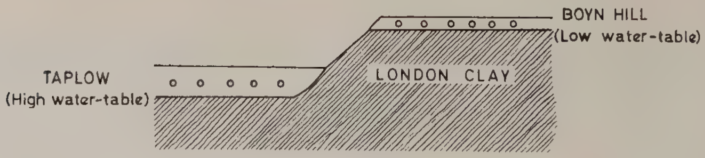


Fig. A

The Taplow terrace would have a high water-table, the Boyn Hill a low one, and water would be absent from the London Clay bluff.

There were thus three different sets of conditions which, as he understood it, would make the interpretation of resistivity data difficult. Bore-holes would be necessary as confirmatory evidence in each of the three parts of the site and he would like to be sure that such factors were taken into account in the interpretation of geophysical data.

---

*W. Bengry and Sons, Ltd., Herefordshire.

†Ministry of Housing and Local Government (Geology Section), London.

Dr. D. B. Morris* asked Mr. Vann if the costs of between £20 and £30 per day for resistivity included the costs of reports. If not, how much extra would a report cost, e.g. for a five-day survey? Generally, the electrode separation was *not* equal to depth of investigation, except in a few cases of uniform conditions. That meant that electrode separations of 100, 200 or 300 ft might be necessary, thus slowing down rates of progress.

Would the new method of interpretation for surveys in Essex (see p. 9) be published and did the problem concern lateral or vertical changes in resistivity?

Resistivity was not generally used at sea, but continuous seismic profiling was widely used. Such devices could produce 20 to 30 miles of bedrock profile per day, which was much quicker and more detailed than results obtained on land.

Mr. J. K. Shanklin,† dealing with Papers 1 and 4, said that he thought that the evaluation of a Bunter Pebble Bed deposit could be tackled better by a geologist than by a geophysicist—the implication being that the use of only one geophysical method would not evaluate a sand and gravel deposit. He thought that at least two methods would be necessary, but those could not be justified on cost grounds. Coupled with that was the fact that the evaluation of the geophysical recordings themselves was a highly skilled task that could only be dealt with by relatively few people.

In general he thought that the back-acter, which could dig a hole 20 ft in depth (usually sufficient to penetrate a level terrace deposit), was a more useful tool than the drilling machine for evaluating the sand and gravel deposits. In the case of glacial deposits it was often possible to dig holes at different levels.

Most deposits were bought at auctions, etc., and were owned by the concern requiring the evaluation. More important than being over-concerned by the temporary extra land disturbances involved in using a back-acter was the effect of obtaining a correct evaluation. Sand

---

*Hunting Surveys, Ltd., London.
†Independent Consultant.

and gravel deposits were so variable that it was almost imperative to examine the material in the ground. If it extended below 20 ft a dragline hole could explore to 50 ft on one lift, using a machine no bigger than, say, a 38 RB.

Mr. P. G. J. Gray* asked the authors of Paper 4 to clarify a point regarding the removal of overburden. Where overburden was greater in depth than the 6 ft limit mentioned on page 62, were scrapers used to that depth and the deeper overburden removed by excavators, or were excavators used from the start?

He would also be grateful for more information on the deposition of fly ash used in land restoration and on the percentage consolidation of the ash allowed for in deciding the level of the original fill.

WRITTEN CONTRIBUTIONS

Mr. G. J. Shrimpton:† In Mr. G. Vann's reference to drilling in unconsolidated beds it is pointed out that water has to be used when employing the lining shell, and that the presence of such water has the disadvantage of washing the material. To overcome this I wonder if it would not be possible to grout a small volume of the material in order to consolidate it prior to sampling, using chemical grouts such as AM-9 and Chrome Lignin, the latter having been used in Derbyshire a few years ago to consolidate dry land.

Further, it is possible that there is an application for the rotary drilling technique employing a double-tube swivel core barrel and face discharge bit in these unconsolidated formations. Small portable drilling machines exist and are capable of core drilling to a depth of some hundred feet, the power unit being a two-stroke petrol engine. The complete equipment is readily dismantled and requires two to three men for its transportation. Such equipment has been used in prospecting operations in the mining industry. It is probable that samples obtained by these techniques will be more costly; however, the advantage of obtaining more accurate information will eventually more than offset any additional cost.

---

*Barnato Brothers, Ltd., London.
†Camborne School of Mines.

Mr. K. W. Harriss:[*] The authors of Paper 4 dispose rather rapidly of the use of the gravel pump as an efficient method of extraction for sand and gravel. The example (p. 63) they give of an 8-in pump delivering up to 300 yd is, to my mind, rather outmoded, there being pumps available commercially which are handling difficult materials through well over 700 yd of pipe.

The efficient modern gravel pump, utilizing the latest materials for construction and capable of heads up to 180 ft, is a far cry from the older unit. With correctly engineered pipelines blockages do not occur and spares costs are often below $\frac{1}{2}$d./yd^3 of gravel pumped.

The authors refer (pp. 64–5) to recent advances in water pump design; their opinions on any investigations into the modern gravel pump would be welcomed.

Mr. E. Rawcliffe:[†] I am sure that engineers engaged in assessing the quality of sand and gravel deposits from samples will agree with Mr. Vann that the first choice in proving deposits would be the standard shell and auger bore-hole, as samples thus recovered indicate changes of stratification more accurately than by most other methods.

Mr. Vann may wish to comment on the following points in bore-hole sampling:

(*a*) As much use of a bore-hole should be made as possible if an accurate assessment of quality is required, or may be required later. Ideally, all the spoil should be retained in suitable containers, but in any case 50-lb samples of each soil type or of each 5-ft depth in the same stratum should be obtained.

(*b*) Large samples of overburden give the engineer an idea of what has to be stripped, taken away and possibly built into stable spoil heaps; large samples of sand and gravel are needed for all the usual laboratory tests for concrete aggregates.

(*c*) The daily logs of the drilling foreman can provide information on the presence of cobbles or boulders in the gravel deposits which might not be apparent from the bulk samples taken.

I would also like to ask the author if it is possible to measure the resistivity of a soil sample by a simple laboratory experiment, and would the result have any practical value in assessing similar deposits in the field, in the same locality?

---

[*] Simonacco, Ltd., Carlisle.

[†] W. and C. French, Ltd., Essex.

Dealing now with Papers 2 and 4, the authors may find the following comparisons, of the cost of sand and gravel produced by modern mechanical means with that produced by purely manual labour, of interest.

Case 1: East Nigeria, 1959—Sand was dug from the Niger River sandbanks by workmen using headpans and standing in several feet of water, loading into dugout canoes. The canoes were than paddled to shore and the sand unloaded by headpan and stacked in heaps on the bank. It took approximately 100 headpans for 1 yd^3 of sand and overall production was about 4 yd^3 per man per day. Wages were 3s. to 4s. per day and the sand was sold for 4s./yd^3, excluding transport.

Case 2: Malaya, 1961—A river 20 ft wide and 2 ft deep brought down sand from the hills and flowed at a rate which just moved the sand along the bottom. Two Chinese dug shallow pits in the river bed and dug out the sand which collected each day. Their joint production was about 10 yd^3 per day and the sand sold at 7s./yd^3, excluding transport.

Case 3: North Nigeria, 1962—Gravel-sized nodules of rock were found in silty soil down to depths of 5 to 10 ft, usually on crests of low domed hills. The gravel was sieved out by hand when the soil was dry, the yield being 10 to 20 per cent of the total volume of excavated soil. The gravel was very dirty, but was otherwise quite satisfactory for concrete making. Output of gravel per man was about $\frac{3}{4}$ yd^3 per day, wages were 4s. per day and the gravel was sold by local contractors at 9s./yd^3, excluding transport.

Mr. M. A. Brooke:[*] I joined the staff of Kamunting Tin Dredging, Ltd., a well known Malayan producer, in 1927, just when the Company was in the process of converting a suction cutter dredge to the normal bucket dredge. The suction cutter had been installed in the 1925–26 period and after a short trial the plant was found to be unsatisfactory and the equipment disposed of.

Although I had no experience with the suction cutter dredge, I naturally discussed with all concerned why the suction cutter dredge was found to be unsatisfactory. I am sure that the reasons given will be of interest, particularly since I had them first-hand from the people who had actually operated the plant:

(1) the plant was under-powered;

(2) the treatment system (sluice boxes) was quite unable to deal with the large and irregular flow of material;

(3) the inability of suction cutters to deal with beds of stiff clay, the

---

[*] Department of National Development, Australian High Commission, London.

large quantity of buried timber found on the property and the occasional large boulders; and

(4) the difficulty of cleaning up the stiff shale bedrock.

The bad design of the first plants used in Malaya should not count against the suction cutter principle but, nevertheless, the other difficulties mentioned were at the time considered to be insurmountable. If present-day hydrocyclones and distribution boxes had been fitted to these plants in 1925, considerably less trouble would have been experienced in the treatment plants.

Large logs (from 20 to 50 ft long) were often encountered on this property, besides many tons of smaller logs and stumps—a problem which apparently could not be handled with the suction pipe in the same way as by the bucket ladder. I was interested to see that Mr. J. C. C. Hill's paper refers to the large tonnage of timber and refuse which was collected on the bows (p. 47) and I wonder if it was floating or sunken.

The inability to deal with logs, clay and boulders, which caused much time to be lost and created difficulties in cleaning bedrock, was the real reason why the suction cutters were considered to be unsuitable for the Kamunting property. Furthermore, even in places where there were no timber or boulder problems, it was found very difficult to clean up bedrock where most of the values were found and even more difficult to check if bedrock had been cleaned up. With bucket dredges the dredgemaster can always 'drop back' and visually inspect the work of his subordinates—not possible with the plants working on the suction principle.

With regard to the possibility of suction cutters being able to clean up the highly valued material between pinnacles, I would agree that the bucket dredge does not do a very good job here and much payable ground is left behind. On the other hand, I can see no reason why the suction cutter would do a better job—in fact I do not think it would do as well and the limestone pinnacles would play havoc with the cutters.

There is undoubtedly a big field for suction cutters and suction dredges in free-running material, particularly where the mineral to be recovered is not highly priced, e.g. in the beach sand industry, dredging for ilmenite, rutile, zircon and the like, but not, in my opinion, where there are large quantities of medium to stiff clay to be worked and where boulders and sunken timber are to be expected. I feel that for a valuable mineral like cassiterite, where it is so essential to clean up bedrock, bucket dredging still remains supreme.

Authors' Replies

(Papers 1–4)

Mr. G. Vann: In answer to Mr. I. H. McLean and Mr. N. Brindle (pp. 78–80), accuracies are normally quoted only when geophysical work has been followed by trial boring giving the essential correlation data. In these circumstances predicted thicknesses are compared with the actual results, thereby giving a guide to the probable margin of error in interpreting the geophysical data over the site as a whole. Naturally, other factors, such as the range of thickness variation, the presence of barren areas (which usually have a larger impact on volumes than minor thickness variations) and of thin clay layers in the gravel sequence which may be apparent in borings but not in the geophysical results, are taken into account.

As pointed out in the paper, some evidence is beginning to become available for areas surveyed twenty years ago which allows comparison of predicted and actual volumes. With the accelerated rate of working deposits it is hoped that such factual data will enable likely tolerances for different types of deposit in various parts of the country to be established.

With regard to Mr. McLean's query on Bunter deposits, our experience has been that neither resistivity nor seismic techniques are likely to detect the presence of pebble horizons in Bunter sandstones. Therefore, even if the pebble layer had been established by drilling, the employment of constant separation traverse resistivity readings in most cases would not enable the lateral extent of the layer to be established.

Mr. J. K. Shanklin's comment (p. 81) on the evaluation of these deposits by a geologist rather than by a geophysicist being better overlooks the fact that geophysics and drilling are both tools to be used by the geologist. If the geologist cannot be aided by geophysics then more boring is likely to be required. Boring is generally more expensive than geophysics and conventional percussion techniques in consolidated rocks tend to be slow and expensive. At the present time down-the-hole airflush drilling is a relatively cheap way of establishing if pebbles are present or not, but there are difficulties in obtaining representative samples and, in particular, in evaluating sand/stone proportion.

Mr. Brindle's finding that the principal Bunter gravel beds are fairly continuous has not been borne out in the areas we have had to examine; the pebble horizons have been found to be lenticular and it is in just this situation that geophysics would be helpful if a reliable technique could be evolved.

Regarding Mr. J. A'C. Bergne's query (p. 79), the extraction of samples from bore-holes and their testing in the laboratory are relatively simple procedures compared to gold and tin sampling, as it is generally only necessary to determine (*a*) the nature of the constituent fragments, i.e. flint, limestone, igneous, etc., (*b*) the distribution of various particle sizes, and (*c*) if deleterious matter such as clay, iron, chalk, organic matter is present or not.

Mr. Brindle's experience of poor-quality material not having been detected by surveys presumably indicates that insufficient boring had been carried out. That highlights that geophysics cannot be used to eliminate test boring, only to supplement it. However, it is our experience that single-flight auger drilling only gives good samples where overburden is thin so that clay is not carried down into the productive layer and, in any case, only the material above the water-table can be sampled. Resistivity surveying is quicker, cheaper and does less damage than auger boring. This statement also applies to Mr. Shanklin's suggestion of using a back-acter. The back-acter suffers from not being able to bottom deep deposits or to penetrate below the water-table, causes considerable damage and is cumbersome and costly to transport from site to site; the general use of a dragline would be even less feasible.

In our experience Mr. Shanklin's statement that most deposits are bought at auctions and owned by the concern requiring the evaluation is not correct. It has been found that the work has to be carried out with the greatest care in order not to upset tenants or owners prior to purchase and speed of executing the work is also very often a dominant factor to meet auction or option deadlines.

The possibility of using grout to consolidate material prior to sampling, mentioned by Mr. G. J. Shrimpton (p. 82), is interesting, but outside the author's experience. Presumably, rotary drilling would be used to core the consolidated material, but the core would need to be at least 4 in. in diameter to be of any value and the portable drilling machines described by Mr. Shrimpton would not be adequate for this purpose.

In answer to Mr. W. Potter (p. 80), rotary coring is not applicable to unconsolidated deposits, but there is the possibility of using reverse-circulation techniques, recently introduced into this country from the Continent and the U.S.A., for water-well drilling.

In the present writer's opinion there would be no justification for more expensive sampling procedures to achieve a marginal increase in accuracy at a few positions, as it is obviously a matter of chance if the results obtained are representative of the whole of the deposit. It would seem more important to maintain the order of sampling accuracy achievable with shell and auger techniques, but to evolve means of obtaining faster, and therefore cheaper, drilling so that more holes can be put down and more samples obtained for the same cost.

It should be noted that the civil engineering code of practice for site investigation drilling requires shell and auger methods to be used for sampling sand and gravel. Although some washing of material can take place by the action of the shell, any appreciable silt or clay content will still be noted by the careful foreman experienced in site investigation requirements.

I agree with the points made by Mr. E. Rawcliffe (p. 83) and in particular with the need for detailed bore-hole logs and the retention of useful size samples. So far as measuring resistivity in the laboratory is concerned, that would have no significance in relation to the field measurements.

I am also in full agreement with Mr. R. O. Warburg (p. 80) on the need for geological control of geophysical work. In the case quoted interpretation of the resistivity data might be difficult if the essential data on water-levels were not available. In many surveys, particularly over river deposits, information on relative topographic levels assists greatly in making a proper assessment of the geological structure from geophysical data; on the other hand, in glacial areas it is very rare for topography and the presence or absence of gravel to be closely related, other than in such obvious features as eskers or drumlins.

In reply to Dr. D. B. Morris (p. 81), the daily charge of £20 to £30 per day quoted for a resistivity survey does not include the costs of a report. For a 5-day survey the author's Company would charge about £25 for a report.

It is the author's experience, based on the order of 10 000 resistivity depth-probe observations at bore-hole positions, that electrode separation is in fact nearly equal to depth of investigation. It is necessary, however, to penetrate considerably below the interface to be determined but, at most, twice the depth is generally adequate; penetrations of more than 100 ft are only exceptionally necessary in gravel surveys.

The new interpretation method developed for surveys in thick overburden areas of Essex will not be published until sufficient surveys have been completed to prove its validity in a variety of conditions.

With regard to Mr. Potter's query, the water-table can be located by resistivity surveys if it is not virtually coincident with a geological interface; a water-level in the middle of a sand and gravel layer would be apparent, but correlation would be necessary to differentiate it from the lithological changes.

So far as lack of contrast in resistivity work is concerned, difficulty can arise if the gravel : overburden ratio falls below 1 : 1, when the higher resistivity of the gravels may be obscured by the clay values.

Additionally, where thick deposits of gravel (say over 50 ft) rest on clays there is often little or no apparent contrast in resistivity values, partly due to the fact that with certain types of clay the resistivity increases with depth of burial, just as seismic velocities tend to increase in the same way.

In conclusion, there is much to be learnt in the execution and interpretation of geophysical surveys for non-metallic minerals and it is to be hoped that, with the increasing awareness of the potentialities of the methods for the sand and gravel industry, time and money will become available for research. A great store of technical information, which could well be used for this purpose, is locked away in files.

Dr. F. Konz: In reply to Mr. R. Whiteside (p. 77), the minimum working depth for the ordinary floating grab dredge lies at about 30 ft. Depths between 15 to 30 ft are being tackled by modern bucket dredges with a rather low top-tumbler and by floating grab dredges of the slewing-crane type, the former being more appropriate for hard-packed material.

The restoration of worked-out areas by filling them with waste ash, excavated materials from building sites, etc., is demanded by the authorities if such material is available at an economical distance from the pits. Legislation is becoming more and more rigorous so far as the industry is concerned, and restoration requirements may often prevent the opening of new pits. The excavated lakes are mostly owned by communities or other authorities and are often available for fishing.

Gravel pit owners recommend the owners of areas excavated, where such areas do not have to be reclaimed, to allow them to be used for amenity purposes, such as swimming, rowing or yachting and, where possible, as camping grounds.

Formerly, and in particular before the last war, the land for gravel pits was mostly owned by the gravel operators. After the war, as a result of tax and capital conditions, it was very difficult for operators to acquire sufficient land. Present-day contracts are therefore being made for a certain area based on a royalty per cubic metre or ton, usually in relation to the existing selling price.

The capacity of barges transporting material to markets lower down the river is between 400 and 1500 tons. The Rhine itself is a free river without locks, but often suffers from low water in summer. Its tributaries, such as the Neckar, Main and Moselle, are canalized and equipped with a considerable number of locks, which also applies to the canals in Northern Germany. River dues only exist in the form of canal charges, not for the Rhine itself. Off-loading of barges is carried out mainly by using cranes with grab buckets. Only in a few cases were special barges with a conveyor system specially fitted into the bottom of the barge in use; an example of their use was between the Grand Canal d'Alsace and Bâle.

In reply to Mr. P. W. Dobson (p. 77), one manufacturer thinks it possible to reach a hoisting speed of 300 ft/min and intends to employ a speed of 400 ft/min in a special case of dredging in unusual depths. The problem of 'washout' of fine material is not too serious since the closed bucket in a good standard of maintenance loses less material than a drag-scraper or similar equipment. Furthermore, Rhine material generally contains such a high percentage of fine material that a modest loss of the material would not endanger its quality.

Bearing in mind ever-increasing wages, etc., in Germany and elsewhere, it seems that the final target in the construction of floating grab dredges must be their complete automation, i.e. that there should be no operator present, provided that the homogeneity of the material to be dredged and the level of the underlying limiting layer permit.

Mr. J. C. C. Hill: Referring to Mr. M. A. Brooke's remarks on the clean-up of irregular bedrock (p. 84), I think that the relative merits of using a cutter suction dredge warrant further investigation in certain circumstances. I do not see why maintenance of the cutter should be any higher than that of a bucket band and the compactness of a cutter head could be of advantage in getting down between pinnacles.

Modern cutter designs with variable speed controls and maximum torque protection have been successfully applied to the excavation of hard coral rock and I would not expect difficulties with a correctly designed rock cutter. (A typical cutter with removable edges, and which can be used with rock picks, is shown in Fig. B.)

I agree, however, that certain types of clay can cause considerable difficulties and are better attacked by the conventional bucket dredges, as are deposits containing a large amount of buried timber.

The hazard caused by timber referred to resulted from surface floating timber. During the preceding period many trees, undermined by the river, were torn from the banks by the first floods together with bushes and refuse. In addition, in the instance cited timber and earth dams many miles up-river were washed away by the torrent. This material carried down by the torrent collected on the bows of the dredge and on the mooring wires.

In order to bring the general information concerning the recovery of minerals from the sea and underwater prospecting up to date, it is

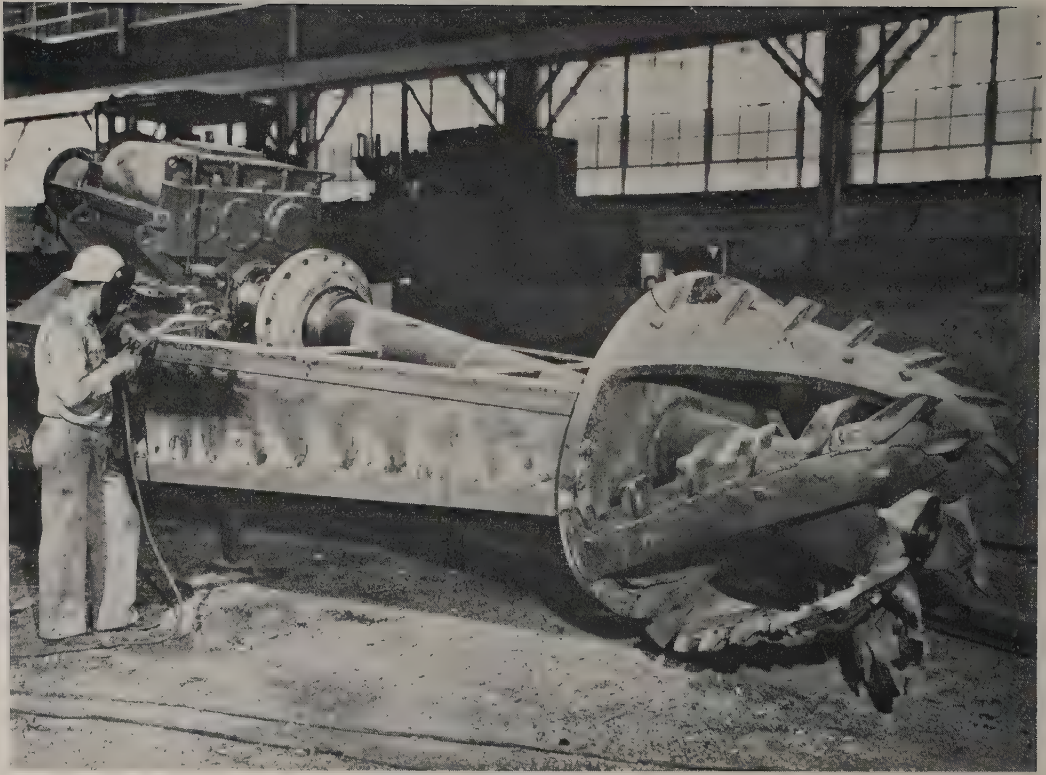


Fig. B

perhaps of interest to note that since the Symposium some development work has been undertaken in connection with the application of air-lifts. The results so far have been encouraging and it is hoped that equipment will be available shortly which will allow 3-ft diameter cased holes to be sunk for prospecting at sea and in rivers. All the connections between the equipment and the vessel will be flexible pipes. In this way it is hoped to reduce the problems of mooring the vessel and allowing for its movement owing to waves and swell.

Cutter suction dredges, floating washing plants and combined systems of dredging for tin have in recent months received renewed interest and are to be installed on various properties. It is noteworthy that construction will shortly commence on a suction hopper dredge for mineral recovery. On this unit primary production will be achieved by a large dredge pump. On the same vessel a series of airlift units are proposed for the clean-up of the bedrock after the main suction pipe has removed most of the deposit.

Mr. R. Whiteside, Mr. T. D. Paterson and Mr. I. H. McLean: We would agree to Mr. J. K. Shanklin's contention that the evaluation of a Bunter Pebble Bed is better tackled by a geologist than a geophysicist. Indeed, the evaluation of any deposit is best carried out by a geologist or mining engineer, who would use the geophysicist's work, where applicable, to make necessary observations amounting to 'simulated boreholes' if, indeed, depth-probe results can be so termed. Often the geophysicist is also a geologist by training.

The universal usefulness of the back-acter or larger excavator in gravel assessment should be disputed for the following reasons.

(1) In cases of high water-table, observations are blind and of necessity imprecise.

(2) Many deposits require deeper excavation than 20 ft.

(3) Usually there is objection on the part of the would-be vendor to the conspicuousness of, and the disturbance caused by, a back-acter or larger machine. If sites are being tested up and down the country four or five at a time, as is the case with this Company, then the use of large machines would not be possible without hiring, which is undesirable from a security standpoint. Also on this score, the opening of large holes invites inspection from the curious and too often sites cannot be tied by a protective option.

(4) A carefully drilled cased percussion drill hole affords ready penetration to 60 or 70 ft under average conditions, giving an adequate sample. It is when the strata to be penetrated are too hard for, say, a Pilcon Wayfarer that a percussive hammer air-flush drill may be

used, giving some doubt to the validity of the sample as regards size distribution. Deep pitting by machine, if possible, might then have to be resorted to.

In reply to Mr. P. G. J. Gray's question regarding overburden removal (p. 82), it might be beneficial to mention that the actual removal of overburden and the type of machinery used are determined by planning conditions, which vary in different parts of the country. Generally, an operator is required to conserve all topsoil for future reclamation purposes and obviously in that case tractors and scrapers are used for taking off the top spit, which is stockpiled for subsequent re-spreading.

In the case of sites where reclamation is proposed by using power-station ash, a certain amount of overburden has to be removed and stockpiled for reclamation purposes and is subsequently re-spread over the fill material. Normally, 1 to 2 ft of the overburden would be removed by tractor and scraper. However, in the case of fairly short hauls it has been found economical to remove greater depths, up to even 4 to 6 ft, by tractors and scrapers, particularly where it is not possible to cast back and, of course, dependent on groundwater conditions.

In the case of reclamation using dry ash or rubbish, some overburden is used to alleviate the dust nuisance. Where, however, there are no over-riding planning conditions affecting the method of extraction, the top spit would be removed by tractors and scrapers and the remaining overburden would be removed by dragline to be cast back into the worked-out area.

There are two means of depositing fly ash for restoration purposes. In some cases, the ash is brought to the excavated area in lorries and tipped directly into the workings, being levelled by a bulldozer. In our experience the material is rapidly consolidated by this method and we have not experienced any noticeable settlement after levelling and re-spreading with topsoil.

Another means of reclamation by fly ash is to pump the ash along a pipeline and discharge it into a worked-out area. Often the worked-out pit is in a river valley and the standing water-level is high. As the pumped material is deposited into the worked-out area, water is displaced and means have to be provided for the water displaced to be discharged. Once the fly ash has displaced the water and has been brought up to the original ground level, it dries out and, in our experience, is quite solid, little settlement taking place. Subsequent spreading of topsoil by dozer therefore presents no difficulty.

With regard to the point about the percentage of consolidation of ash in deciding the level of the original fill, we would say that between 5 and 10 per cent of consolidation should be allowed for. Often, of course, the final level will be determined by the quantity of fly ash available. It may be that there is insufficient material to bring the site back to its original level, in which case it can be graded. In some cases, where there is an excess of filling material, it may be necessary to crown the finished area.

Mr. K. W. Harriss has accused us of disposing rather rapidly of the use of the gravel pump as an efficient method of extraction for sand and gravel (p. 83) and asks if we could express an opinion on any investigations of a modern gravel pump. The Paper was limited to a discussion about our sand and gravel operations in the Trent Valley. Because of the nature of the deposit in these pits other methods of extraction are being used. We cannot, therefore, comment on the recent advances which Mr. Harriss refers to in the design of sand and gravel pumps. All we can say, from our experience generally, is that the equipment we have used in the past has been inefficient, particularly owing to the wear on moving parts and because of blockages in the pipeline and, especially in shallow deposits, owing to the inconvenience of moving the pipelines about. While we appreciate that advances have been made which provide for a considerably longer life of gravel pumps, we are not in a position to comment on these.

GROUP I—ALLUVIAL DEPOSITS
(including sand and gravel)

Session 2—Alluvials (Papers 5–9)

Chairman: Mr. D. S. Burwood *Rapporteur:* Mr. J. E. Denyer

622.013.3:622.345.7(595)

Paper 5

Valuation of Alluvial Tin Deposits in Malaya
with special reference to exploitation by dredging

J. K. BROADHURST, A.C.S.M., A.M.I.M.M.
D. J. BATZER, B.Sc., A.R.S.M., A.M.I.M.M.
Head and Assistant respectively, Field Engineering Department, Anglo-Oriental (Malaya), Ltd.

SYNOPSIS

The current situation regarding valuation and tin recovery in Malaya is summarized.

Present methods used by Anglo-Oriental (Malaya), Ltd., for core volume measurement in virgin ground and tailings are discussed and results of comparative tests are tabulated. Reference is made to the recent introduction of screening of bore samples before *dulang* washing to recover the free cassiterite. With suitable precautions this method gives a good recovery between 14 and 300 mesh B.S.S. The assessment of tailings losses is also made.

A basis for the statistical analysis of results of bores drilled in Malayan alluvial cassiterite deposits, on which 'confidence limits' for the mean bore value may be determined, is suggested, using ln of the bore value. Some typical examples are given. Density of bore grid is mentioned and a comparison of plant recovery against bore value discussed.

Total tin content of typical alluvium is referred to with regard to −300-mesh B.S.S. cassiterite. Factors affecting valuation, apart from the reliability of the individual bore, are mentioned.

It is concluded that the principal uncertainty in valuation at the present time is core measurement and that in other respects existing methods are satisfactory provided that proper care and control are exercised.

New developments both in equipment and technique in boring practice in Malaya have, for many years, been few. With the grade of ore being mined, and the equipment available for mining, ore reserves have been determined reasonably accurately by methods detailed by Harrison.[1] Since 1945 over one thousand million cubic yards have been mined by the group of companies under the management of Anglo-Oriental (Malaya), Ltd., from which the tin concentrate recovered has been 90 per cent of that estimated for the ground actually mined, before allowing for plant losses. These figures are derived from 52 areas mined or being mined with an average grade of 0·37 kati/yd³.* Twelve of these areas show plant recoveries in excess of 100 per cent, in most cases the reasons for the undervaluations being known.

---

* 1 kati = $1\frac{1}{3}$ lb.

In recent years there has been a steady reduction in the grade of alluvium being worked—a trend that will most probably continue. It is therefore becoming increasingly necessary for the lower-grade ground to be valued as accurately as possible.

Many factors affect valuation, but the first consideration is the accuracy of the individual bore, which depends principally on core measurement and recovery of the free cassiterite. Although depth of barren overburden, often as much as 50 per cent in areas currently being valued for dredging, will reduce the effect of all errors, i.e. in weighing, loss of cassiterite and core measurement, these errors have a proportionately greater effect on the shallow lower-value bores.

CORE MEASUREMENT

Virgin Ground

The standard practice of the authors' Company is to measure the volume of large stones, stiff clay or tough sandy clay—recovered from the bore more or less intact—by water displacement. Small stones and the disintegrated portion of the core are measured by ramming the material thoroughly into the measuring bucket with no allowance for swell. The volume of loose and very wet samples is adjusted by the use of the graph as described by Harrison,[1] a simplified form of which is shown in Table I, the object being to compact the sample to the same density as it occurs *in situ*. The achievement of this aim depends to some extent upon the judgement of the engineer in charge.

A method of core measurement recently used in Nigeria[2] employs a hydrometer for measuring the slime content, the balance of the sample being determined by water displacement. Check measurements by this method have been made in Malaya and gave consistently smaller volumes, with consequent higher valuation, as compared with the procedure described above.

Ground in Malayan alluvials can vary considerably and may sometimes be very 'loose' with a high water content approaching a quicksand. Ingham and Bradford have described the young alluvium in the Kinta Valley as 'consisting chiefly of unconsolidated sand and gravel',[3] whereas the old alluvium close to the granite masses 'has a superficial resemblance to the weathered granite'.[4]

Many attempts have been made to arrive at accurate core measurement, but it is generally considered that the method described is the one best suited to conditions in Malaya. Some verification of its accuracy has been obtained by the use of a light power-drill which enables the casing to be driven and a check valuation calculated by 'pipe factor'. This method of bore valuation is based on the theoretical

TABLE I.—*Proportion of solids in wet ground samples*

The intersection of the column for (1) with the line for (2) gives the percentage of solids applicable.

Assumed weights of alluvium: $\frac{1}{4}$ ft^3 = minimum of 20 kati, maximum of 25 kati.

(2)	12·0	12·5	13·0	13·5	14·0	14·5	15·0	15·5	16·0	16·5	17·0	17·5	18·0	18·5	19·0	19·5	20·0	20·5	21·0	21·5	22·0	22·5	23·0	23·5	24·0	24·5	25·0	25·5
(1)																												
19	4	11	18	25	31	38	45	52	59	66	73	80	86	93														
19·5	4	10	17	24	30	36	43	49	55	62	68	75	81	87	94													
20	4	10	16	23	28	34	40	46	52	59	64	71	76	82	88	94												
20·5	4	9	15	21	26	32	38	43	49	55	61	67	72	78	83	89	95											
21	3	8	14	20	25	29	36	41	47	52	57	63	68	73	78	84	89	95										
21·5	3	8	13	18	24	29	34	39	44	49	54	60	65	70	75	80	85	90	95									
22	3	8	13	18	22	27	32	37	42	47	52	57	62	66	71	76	81	86	91	96								
22·5	3	7	12	16	21	25	31	35	40	45	49	54	58	63	68	73	77	82	86	91	96							
23	2	7	11	16	20	25	29	34	38	43	47	52	56	60	65	69	74	78	82	87	91	96						
23·5	2	7	11	15	19	23	28	32	36	41	45	49	54	58	62	66	70	75	79	83	87	92	96					
24	2	6	10	15	19	23	27	31	35	39	43	47	51	55	59	64	67	72	76	80	84	88	92	96				
24·5	2	6	10	14	18	22	26	30	34	37	41	45	49	53	57	61	65	69	73	77	80	84	88	92	96			
25	2	6	10	14	17	21	25	29	32	36	40	44	47	51	55	59	62	66	70	73	77	81	85	88	92	96		
25·5	2	6	9	13	17	20	24	27	31	35	38	42	46	49	53	57	60	64	67	71	75	78	82	85	89	93	96	
26	2	5	9	12	16	19	23	26	30	33	37	41	44	47	51	55	58	61	65	68	72	76	79	82	86	89	93	96
(2)	12·0	12·5	13·0	13·5	14·0	14·5	15·0	15·5	16·0	16·5	17·0	17·5	18·0	18·5	19·0	19·5	20·0	20·5	21·0	21·5	22·0	22·5	23·0	23·5	24·0	24·5	25·0	25·5

(1) *Wt, kati/$\frac{1}{4}$ ft^3 of solids*

(2) *Wt, kati/$\frac{1}{4}$ ft^3 of slimes or wet sample*

volume of material cut by the casing cutting shoe and is normally used in gold valuation. The volume for 1 ft of depth, usually calculated on 88 per cent of the cutting-shoe area, is known as the 'Radford Factor'.

In hand boring in Malaya the casing is continually being turned without driving ahead and bore valuation by the pipe factor method proved unsatisfactory and has long been discarded. With a powered churn-drill, however, the usual practice is not to turn the pipes but to drive them ahead and then pump out the core. By measuring the core rise in the casing after driving an additional check is provided on the volume cut.

After pumping, the sample obtained can also be measured in the usual manner. Swell and compaction do not enter the calculation using a pipe factor.

TABLE II.—*Valuation by two different methods using power drill*
(Virgin ground throughout)

Area	No. of bores	Average depth, ft	Average volume of sample per ft, tenths of $\frac{1}{4}$ ft^3	Core recovery per cent	Average value kati/yd^3 high value bores cut to 1 kati/yd^3 (72% Sn)	
					Pipe factor method	Bucket volume measurement and ft kati calculation
1	2	96·5	5·7	95	0·21	0·24
2	6	50·5	4·9	82	0·38	0·39
3	10	55·0	5·2	88	0·11	0·11
4	1	150·0	5·5	92	0·02	0·02
5	7	63·0	4·9	84	0·47	0·43
Average:			5·1	86	0·26	0·26

Table II gives a comparison of valuation calculated by the two different methods using a light churn-drill, a close measure of agreement being revealed.

Tailings

It has generally been found that when values of bores in tailings have been taken at their face value dredge recoveries have been of the order of 65–75 per cent. A certain plant loss is inevitable and probably amounts to between 10 and 15 per cent. The additional 10–25 per cent can be explained as follows:

When material is excavated expansion takes place as illustrated below:

Material	*Original bulk*	*After excavation expansion to*
Earth	100	115–125
Sand and gravel	100	115–125
Clay	100	approx. 130

Alluvial ground in Malaya often has a fair proportion of clay, more particularly as barren overburden, and estimates indicate that swell up to between 15 and 25 per cent takes place when it is redeposited in the form of dredge tailings. Some tailings bores have shown an appreciable increase in depth as compared with the original alluvium, in spite of the removal of some silt and slime to settlement areas elsewhere.

In boring tailings looseness of the ground is usually apparent and it is frequently necessary to hold back the bore casing to prevent it sinking under its own weight. If the bore sample is compacted in the sample bucket and measured in exactly the same manner as a sample of virgin ground, overvaluation takes place. For example, if 100 yd^3 of virgin ground contain one picul* of concentrates it would have a value of one kati/yd^3. If this were excavated without treatment it could be expanded in the form of tailings to about 120 yd^3 and would still contain one picul of concentrates, giving a value of 0·83 kati/yd^3, which would be the correct value of the tailings. If this were compacted to 100 yd^3 (which is what tends to occur in the bore-sample measuring bucket) it would still contain one picul of concentrates, giving the original value of 1·00 kati/yd^3. To convert this value to the correct value of 0·83 kati/yd^3 an adjustment of 83 per cent is necessary. It will be appreciated that tailings will contain the original earth and barren clay overburden mixed in with them.

Core recovery for Malayan hand sets (cutting-shoe diameter $4\frac{11}{16}$ in.) in virgin ground generally averages 65–68 per cent, whereas core recovery in dredge tailings is usually between 55 and 57 per cent, i.e. about 85 per cent of the core recovery in virgin ground—which tends to confirm that swell has taken place in the tailings.

Two tailings areas were bored using a light power-drill and the value arrived at (*a*) by pipe factor, which eliminates the effect of swell and compaction, and (*b*) by the ordinary method of bucket volume measurement and foot kati calculation with an 80 per cent factor applied to the tailings sections of the bores. Agreement in valuation was within 0·01 and 0·05 kati/yd^3 respectively. The authors' Company has for some years applied a 'tailings factor' to those sections of bores which are in tailings, usually of 80 per cent. Core recovery gives a guide to the actual 'factor' to be adopted.

---

* 100 katis = 1 picul.

RECOVERY OF FREE CASSITERITE IN THE BORE SAMPLE

The sources of loss of free cassiterite in the bore sample are (*a*) the decanted slimes; (*b*) the *dulang** tailings; and (*c*) the tin dressers' tailings. That *dulang* loss occurred has been appreciated for some time, particularly in the coarser ranges (+14 mesh B.S.S.), as such particles tend to roll off the *dulang*. Few areas currently being valued for dredging contain appreciable amounts of coarse cassiterite, but where it does occur some allowance should be made.

Tests have been carried out on bore samples during the past few years, first by passing decanted slimes through a pressure filter and vanning representative samples, but more recently—described by Williams[2]—by pumping the decanted slimes by a Monopump to a 3-in cyclone, the overflow of which all passes 300 mesh B.S.S. The underflow and *dulang* tailings were treated on a half-size table, the concentrates being upgraded by employing a superpanner and hand vanning. All tailings were carefully checked by vanning to ensure that they contained no further recoverable cassiterite.

It was found that by screening the bore sample at site, after puddling, into $\frac{1}{4}$-in, $\frac{1}{8}$-in, 20-, 40- and 60-mesh nominal fractions, and by washing the different sizes separately on the *dulang*, that *dulang* loss was reduced to negligible amounts—in the range $-14 + 300$ mesh B.S.S. Dredge plants cannot be expected to save appreciable amounts of cassiterite in the -300-mesh B.S.S. fraction. *Dulang* losses amounted to about 10 per cent prior to the adoption of screening as routine practice, and the screening has also simplified the work of the tin dresser.

With samples already screened at the bore site it has proved advantageous, in taking spot checks, to pass the +40-mesh fraction through a small laboratory jig and treat the -40-mesh fraction on a half-size sand table. In this way fewer passes are required to recover all the free cassiterite.

Checks on the tailing products of individual bores in different areas have been taken, Table III giving some typical results.

---

* Flat wooden pan used in washing alluvial ore-bearing sands.

TABLE III

Bore no.	*AG* 11			*S* 15			*AC* 39			*AQ* 272		
Depth, ft	140			60			100			23		
Bedrock	Unbottomed			Weathered granite			Unbottomed			Weathered shale		
Barren overburden, ft	17			37			33			14		
Value field concentrates, kati/yd^3	0·36			0·09			0·16			0·36		
Assay adjustment factor*	77 per cent			113 per cent			114 per cent			103 per cent		
Value (concs. 72 per cent Sn) kati/yd^3	0·28 (0·277)			0·10(2)			0·18(2)			0·37(1)		
Total wt of concs. (72 per cent Sn), g*	67·94			14·24			38·81			16·20		
Free Recoverable Cassiterite in 'Barren Overburden' (concs., g 72 per cent Sn)												
Mesh B.S.S.	+300	−300	Total	+300	−300	Total	+300	−300	Total	+300	−300	Total
Combined loss from *dulang* tailings, decanted slimes and tin dressers' tailings	0·307	0·122	0·429	0·370	0·467	0·837	0·009	0·028	0·037	not measured		
Approx. overall value, kati/yd^3	0·02			0·01			0·007			—		
Loss of Free Recoverable Cassiterite in Tin-bearing Ground (concs., g 72 per cent Sn)												
Dulang tailings	0·130	0·303	0·433	0·089	0·397	0·486	0·166	0·156	0·322	0·172	0·055	0·227
Decanted slime	0·376	0·430	0·806	0·079	0·272	0·351	0·077	0·077	0·154	0·021	0·007	0·028
Tin dressers' tailings	1·999	0·055	2·054	0·082	0·214	0·296	0·418	0·054	0·472	0·070	0·040	0·110
Total	2·505	0·788	3·293	0·250	0·883	1·133	0·661	0·287	0·948	0·263	0·102	0·365
Amended adjustment factor† (if +300-mesh fraction incl.)	79 per cent			115 per cent			116 per cent			105 per cent		
Amended value, kati/yd^3 (72 per cent Sn)	0·28(4)			0·10(4)			0·19 (0·186)			0·38 (0·378)		
Undervaluation, kati/yd^3 (72 per cent Sn)	0·00(7)			0·00(2)			0·00(4)			0·00(7)		

* See Table IV. † See text following Table IV.

WEIGHT OF CONCENTRATE AND ASSAY ADJUSTMENT

Concentrates and *amang** from individual bores are generally grouped, for assay and recovery tests on the *amang*, in order to adjust the bore values to terms of a common metallic tin content, usually 72 per cent Sn. It has been observed that the concentrates weighed precisely on the laboratory balance are generally in excess of the 'field' weight by an amount within the range of the accuracy of the field balance (about 2–3 per cent on a typical Vallentine scale[1]), as there is a tendency to be conservative when weighing in the field. When the field weight exceeds the accurate weight by more than 3 per cent the cause should, if possible, be ascertained. It can often be explained by loss in packing and transport. The apparent loss or gain is often greater than the combined loss of cassiterite in dressing the sample, that is from the *dulang* tailings, decanted slimes and tin dressers'

TABLE IV.—*Assay certificate*

Bore *AG* 11	Field concentrates 0·20 kati and over	
	Assay per cent Sn	Wt g
(a_1) Field weight of concentrates		88·76
(*a*) Laboratory weight of field concentrates		92·00
(*b*) Assay of field concentrates per cent Sn	52·8	
(*c*) Equivalent weight of 72 per cent Sn conc. $= \frac{a \times b}{72}$	72	67·47
(*d*) Assay and weight of total *amang*	0·14	517·50
(*e*) Weight of *amang* taken for magnetic separation		300·00
Weight of magnetics		122·00
Weight of non-magnetics		178·00
(*f*) Weight of final concentrates from retreatment non-magnetics		0·31
(*g*) Hence weight of tin concentrate recoverable from total *amang* $= \frac{f \times d}{e}$		0·53
(*h*) Assay of tin concentrate (*f*) recovered from *amang*	63·5	
(*i*) Equivalent total weight 72 per cent Sn conc. recoverable from total *amang* $= \frac{g \times h}{72}$	72	0·47
(*j*) Total equivalent weight of all conc. @ 72 per cent = $c + i$	72	67·94
(*k*) Assay adjustment factor $= \frac{j \times 100}{a_1}$		76·54 per cent
	say	77 per cent

---

* Heavy minerals discarded in the process of tin-ore concentration.

tailings. More precise field balances would obviate this error, but the Vallentine scale is a handy instrument for field use and weighing error can be compensated when calculating the assay adjustment factor as shown in the *pro forma* assay certificate (Table IV).

Example of adjustment of bore value:

Bore *AG* 11 value 0·36 kati/yd^3 (field concs.)
0·36 × 77% = 0·28 (0·277) kati/yd^3 72% Sn.

The 'amended assay adjustment' factor for bore *AG* 11, as shown in Table III, is determined as follows:

Total equivalent weight of all concentrates (72 per cent Sn)	67·94 g
Add +300-mesh cassiterite from *dulang* tailing, decanted slimes and tin dressers' tailings in tin-bearing ground	2·51 g
Total	70·45 g

$$\text{Amended assay adjustment factor} = \frac{70{\cdot}45 \times 100}{88{\cdot}76} = 79 \text{ per cent}$$

Bore-value field concs. 0·36 kati/yd^3 × 79 per cent = 0·28(4) kati/yd^3 72 per cent Sn

It appears that (*a*) the amount of free cassiterite, including the −300-mesh B.S.S. fraction in the 'overburden', is insignificant and (*b*) loss of +300-mesh B.S.S. free cassiterite in *dulang* tailings rarely exceeds 0·30 g and that the combined loss, including decanted slimes and tin dressers' tailings, need not exceed 1·50 g of cassiterite. As the effect on the assay adjustment factor is proportionate to the total amount of cassiterite in the bore, only low-grade shallow bores are likely to be affected and then by an amount not exceeding 0·02 kati/yd^3 (see bore *AQ* 272, Table III).

The losses in tin dressers' tailings are the least constant and occasionally are higher than the other losses. The bulk of these tailings is small and therefore they can conveniently be brought in from the field for retreatment if necessary. In Malaya, because of the nature of the terrain, i.e. swampy jungle and the distances involved, samples are usually washed at the bore site.

In the past bores for assay and recovery tests have generally been combined into two groups—of 0·20 kati/yd^3 and over, and under 0·20 kati/yd^3.

Table V shows the results of eight bores assayed separately and a calculated combined assay.

Low-value bores generally have a higher *amang*: cassiterite ratio than high-value bores and, if the tin dresser has washed the bore concentrate to a high assay, much cassiterite may have been washed into the *amang* and when the cassiterite is recovered from the *amang* in the laboratory, by magnetic and high-tension separation, a high assay

TABLE V

Bore no.	Depth ft	Field value kati/yd^3	Individual assay		Combined assay		Combined assay omitting bores 0·05 kati/yd^3 and under	
			A.A.F. per cent	Value 72 per cent Sn kati/yd^3	A.A.F. per cent	Value 72 per cent Sn kati/yd^3	A.A.F. per cent	Value 72 per cent Sn kati/yd^3
1	60	0·27	110	0·30	109	0·29	108	0·29
2	56	0·16	99	0·16	109	0·17	108	0·17
3	59	0·62	107	0·66	109	0·68	108	0·67
4	52	0·14	115	0·16	109	0·15	108	0·15
5	51	0·16	105	0·17	109	0·17	108	0·17
6	56	0·08	111	0·09	109	0·09	108	0·09
7	51	0·03	166	0·05	109	0·03	—	—
8	49	0·03	158	0·05	109	0·03	—	—
Total over/undervaluation as compared with individual assay						−0·03		0·00

adjustment factor may result. It will be seen that if bores 7 and 8 are omitted (Table V) the combined assay adjustment factor is 108 per cent and the net effect is to give no change in overall value. Bores of value 0·05 and below are now grouped for assay separately.

TOTAL TIN CONTENT OF ALLUVIUM

Apart from the tests already referred to, another investigation has been carried out to ascertain total tin metal losses in bore samples. Bore samples (from which the decanted slime was pressure filtered) were dried and split accurately in the field, one portion being used for routine bore valuation and the other being screened and the total tin content of each mesh fraction determined by chemical assay.

Total tin metal losses were ascertained in general to be 40 per cent, of which 10–20 per cent was in the *dulang* tailings (before screening was introduced). Of this 50–85 per cent was recoverable as free cassiterite.

About 20 per cent of the total metal content was in the decanted slimes, of which 60–90 per cent was recoverable as free cassiterite by vanning assay. Of this free cassiterite about 90 per cent was in the −300-mesh B.S.S. fraction. These tests also showed that the total tin in the −200-+300-mesh B.S.S. fraction of the ground sample was approximately 2–5 per cent. It is likely, however, that of the total free cassiterite in the ground a considerable amount is in the −300-mesh B.S.S. fraction, a large proportion of which is probably much finer than 300 mesh. Little of this usually appears in screen analyses of bore concentrates, being lost in the decanted slimes.

Chemical assays for determining the tin content of very low-grade ore, as encountered in alluvial ground, can be misleading. Some tin is undoubtedly in chemical or physical lock.

BORE GRID

In some alluvial deposits in Malaya and South Thailand the natural logarithm of the bore values follows an approximately normal distribution.

The ln distribution of the values was established by the well known method of comparing the observed frequencies of the ln of the bore values in consecutive limited classes, with the expected frequencies at the class mid-points calculated from the appropriate normal curve for the data. The χ^2 test is then applied to measure the 'goodness of fit' of the two sets of frequencies.[5] A histogram of the observed frequencies, with the appropriate normal curve superimposed for a typical case, is given in Fig. 1.

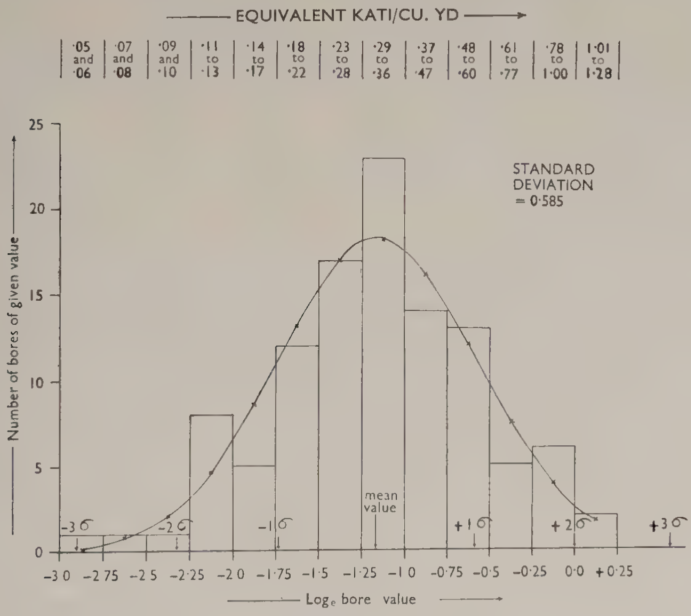


Fig. 1

In these frequency distributions only those parts of an alluvial area of the same general nature have been used in any one analysis. For example a selection of low average value has not been mixed with an area of high average value. Similarly, selected areas of fairly consistent and of erratic values need to be considered separately. In other words the bores should, as far as possible, be members of one 'population'. This necessary restriction is no disadvantage since usually only fairly limited areas are under consideration at any one time.

On the basis of the ln distribution a form of 'confidence limit' may be calculated which appears to give a rough guide as to whether or not the bore grid in any particular case is sufficiently dense for a reliable valuation, with the reservations mentioned below. If the number of bores is less than 30, Student's *t*-distribution is used.[5]

In Table VI the '90 per cent confidence limits' in several areas are given together with the number of bores used, density of bore grid and standard deviation, which gives a measure of how erratic the values are.

TABLE VI

Area	Weighted av. value and 90 per cent confidence limit kati/yd^3	Lower limit as percentage of upper limit	No. of bores	Area per bore acres	Standard deviation (ln scale)
A	0·39 ± 0·03	88	78	7·7	0·532
B	0·26 ± 0·02	86	95	6·7	0·436
C	0·16 ± 0·01	88	116	6·1	0·346
D	0·34 ± 0·03	84	107	3·3	0·585
E	0·50 ± 0·05	82	205	0·9	1·130
F	0·45 ± 0·08	70	63	2·6	1·015
G	0·30 ± 0·04	76	14	5·8	0·278

There is a probability of 0·9 that the average value of the bores may be in error by as much as the upper or lower limits purely by the operation of chance, however carefully the individual bores may have been valued. If the individual bore values are consistently erroneous, owing to some fault in technique or equipment, this presumably would not be disclosed. Meticulous care is therefore essential in the actual boring operation. Even when every care is exercised, however, it is apparent that the possible error in valuation due to chance alone may be larger than that due to losses of cassiterite in washing the samples.

All calculations have been performed in terms of ln bore value. The mean of the distribution is not the same as the weighted average value used in calculating the cassiterite content and the ore reserves, although it may be fairly close. In the examples given in Table VI the ln mean value was used to calculate the 'confidence limits'. If the ln of the average weighted value were used instead the results would be the same, to two places of decimals, in most cases. The standard deviations listed in Table VI are on the ln scale because this quantity clearly has no fixed value in terms of kati/yd^3.

In a normal distribution about two-thirds of the values lie within one standard deviation on either side of the mean value, about 95 per cent lie within two standard deviations of the mean and 99·7 per cent within three standard deviations of the mean. It is therefore considered that all bores of value greater than 2 standard deviations above the mean should be regarded as erratic 'highs' and cut back to that value (i.e. $m + 2\sigma$), where m is the mean value of the distribution.

It would appear from Table VI that 'confidence limits' depend on (1) the total number of bores; (2) the area per bore (density of bore grid); (3) the standard deviation; and (4) the mean value. If the lower 'confidence limit' of the value is expressed as a percentage of the upper limit the effect of magnitude of the value is eliminated.

The standard deviation gives a measure of the 'patchiness' of the deposit, being larger in erratic deposits than in deposits of similar average but more consistent values. To some extent the standard deviation is dependent on the average value of the bores, areas of lower average grade tending to have smaller standard deviations. This is clearly illustrated in Table VI. If area *A* is compared with area *F* of similar value it will be seen that despite a much lower bore density in area *A* the 'confidence limits' in the former are much narrower than in the latter because of the greater consistency of the values in area *A*. As the standard deviation increases the density of the bore grid necessary to attain any particular 'confidence limit' increases rapidly (compare also areas *A* and *E*).

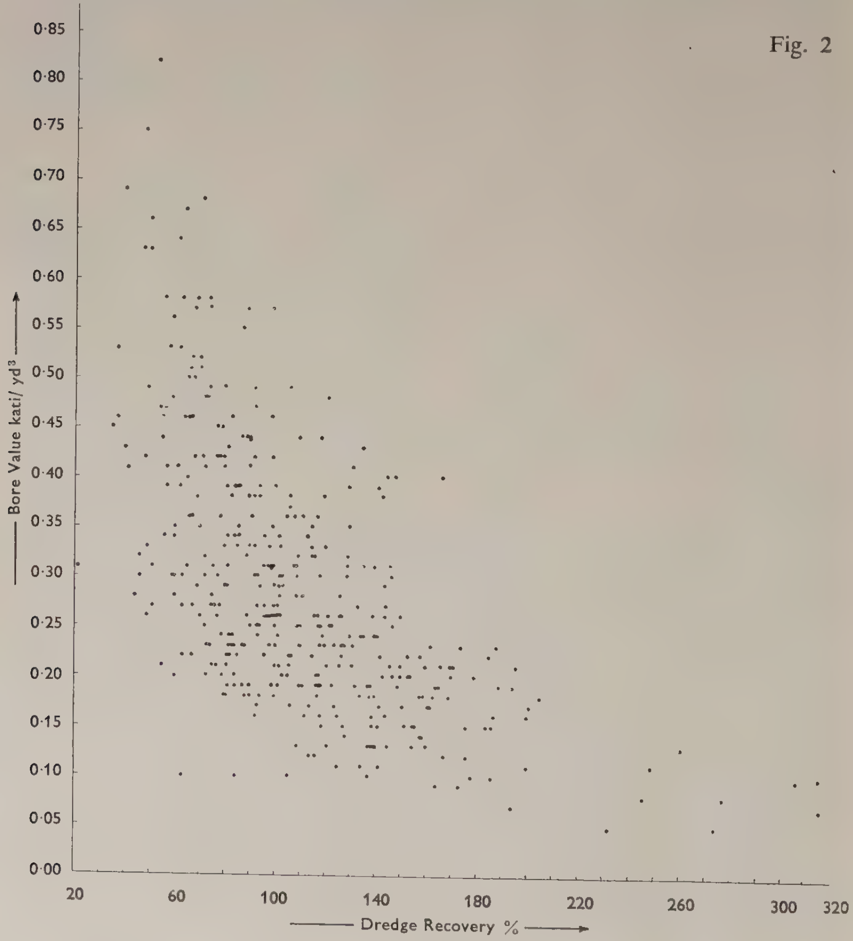
It has been said that low-grade ground needs closer boring for reliable valuation than high-grade ground, but this is not necessarily true, the determining factor being the standard deviation.

If the '90 per cent confidence limits' are wide this may indicate (1) too few bores; (2) bore spacing too wide; (3) unusually erratic ground; (4) some bores unbottomed or unreliable in some other respect (if all bores are in error in the same way, however, this may not show up); and/or (5) salting of bores, deliberate or otherwise, unless this has been done in such a way as to maintain a normal distribution.

It should be pointed out that even though sufficient bores may have been drilled to give a valuation to the desired confidence limits, some additional bores may be required to define more closely the limits of payable ground in particular circumstances. Low-grade ground may contain small relatively high-grade concentrations which may be missed, giving a false impression of consistency. In such ground some parts should be drilled on a much closer grid than normal to test this possibility.

Statistics may be misleading and the method of appraisal described should not be applied or interpreted mechanically. Its purpose is to serve as an assistance to reliable valuation with a minimum of bores.

A tendency to overvalue higher-grade ground and to undervalue lower-grade ground in the past is apparent. This opinion is based on the data presented in Fig. 2 where monthly figures for dredge recovery have been plotted against bore value. It must be stated at once that such a figure for any individual month means very little as it frequently happens that there are only one or two bores in the ground dredged in any one



month. The distribution of nearly 400 such points, shown in Fig. 2, can, however, hardly be due to chance. The data in Fig. 2 are derived from mining by bucket dredges of comparable efficiency on properties of shallow barren overburden, relatively shallow free tin-bearing ground and with weathered bedrock of fairly even contour. The relevant bores were drilled prior to the introduction of screening of bore samples before *dulang* washing.

Usually the higher recovery on the low-grade parts of an area has balanced the lower recovery on the high-grade parts, giving an overall recovery of about 90 per cent by dredging. The variation in dredge outputs from month to month is proportionately smaller than the variation in bore value and there is therefore a tendency for apparent undervaluation of low-grade patches and a similar tendency for apparent overvaluation of high-grade patches. Where fairly extensive areas of exclusively low-grade ground have been worked, however, a consistently high recovery has often been achieved. This has generally been near the end of a dredge's life and outside selected ore reserves involving no risk, as the dredge could be closed down as soon as the ground became unpayable.

The full reason for the apparent overvaluation of bores in richer ground is not known. All bores must be undervalued to some extent as a result of the various losses involved in treatment and washing of the bore samples. In lower-grade ground these losses may be relatively greater, especially in shallow ground where the total weight of concentrate recovered from a bore is small. Furthermore, the lower-grade parts of a selection may contain a few trace or very low-value bores, which have been grossly undervalued on a percentage basis, and also small relatively high-grade erratic concentrations which may not have shown up if the bore grid was rather wide. These factors all work towards undervaluation and although the full explanation is yet to be found the trend to progressive undervaluation in lower-grade ground is quite apparent.

OTHER FACTORS

Apart from the accuracy of the individual bore and the density of the bore grid, the following factors can also influence the valuation of an alluvial property, depending on the method of mining employed.

Incidence of boulders and large gravels.—Boulders and large gravels must be broken up to enter the bore casing or a larger casing has to be used to prevent gross overvaluation and volume calculation may also be affected. Pit digging may be desirable in some cases.

Incidence of clay in tin-bearing ground.—When boring in clayey tin-bearing ground an on-the-spot appraisal of the extent to which the material will break up and release its cassiterite has to be made and recorded on the bore log sheet. This is sometimes very difficult, especially if there is no mining in the immediate vicinity to serve as a guide. Nevertheless, estimation of the proportion of cassiterite in clay is of vital importance in valuation. The installation of special clay-treatment plant may be necessary, although the high running costs and small throughput of such plants makes them inapplicable to low-grade ground at the current cost : price ratio.

Type of bedrock.—Loss of cassiterite in dredging occurs because of spillage and scattering when cleaning up, even under the most favourable conditions of gently contoured weathered bedrock. In limestone bedrock, where there may be numerous pinnacles, the problem is greatly aggravated. In some instances effective dredging depth has been as much as 17 ft less than the average bore depth. It might be best in such cases to use large-scale opencast methods of mining if surface conditions permit.

COSTS

The cost of prospecting and valuation of alluvial areas in Malaya can vary between wide limits, depending on such factors as access, type of terrain, bore spacing, number of bores and depth of the alluvium. Since alluvial areas in Malaya are generally swampy, access for a power-drill will be costly and hand-drilling sets are usually employed. With these the cost per foot increases rapidly below 80 ft, but is generally less than a power-drill to that depth, even in areas of hard standing where a power-drill has easy access. Boring costs conform in general to the following pattern:

	%
Drawing-office charges	2
Surveys	5
Bridging and platforms*	3
Transport	15
Camps	5
Boring labour	33
Supervision	27
Sundries (assays, etc.)	10
	100

* Single-pole footbridge between bore sites and platforms at bore sites in swamp.

Close boring at an established mine is usually the cheapest and costs range from about M$5/- to M$10/- per foot, depending on depth of ground and prevailing surface conditions.† Scout boring in some inaccessible swamp is the most expensive and costs range from about

---

† M$1/- = 2s. 4d. sterling.

M$10/– to M$25/– per foot, depending mainly on accessibility and number of bores drilled.

Power-drills are seldom used because of the nature of the terrain, but they are employed on areas where the ground is firm and easily accessible and can be mounted on pontoons for sea boring in deep water. A small power-drill ($5\frac{1}{4}$-in diameter cutting-shoe) costs about M$10/– to M$12/– per foot and is fairly consistent from surface to bedrock.

Tests are currently being carried out using light rods and it is understood that experiments are also being conducted in jet drilling, at least through low-grade or barren overburden.

CONCLUSIONS

There is probably some scope for improvement in the technique and reliability of the methods of valuation described and much yet to be understood about the process in detail. The main uncertainty concerns the measurement of core volume, but current methods, used with care under proper supervision, are capable of giving reliable estimates of the recoverable metal content of stanniferous alluvial deposits.

Acknowledgement.—The authors wish to express their thanks to Anglo-Oriental (Malaya), Ltd., for permission to publish the data presented.

REFERENCES

1. Harrison, H. L. H. *Valuation of alluvial deposits, 2nd ed.* (London: Mining Publications, Ltd., 1954), 308p.

2. Williams, F. A. Recovery of fine alluvial cassiterite: correlation of bore valuations with plant-scale recovery. *Trans. Instn Min. Metall.*, **70**, 1960–61 (*Bull. Instn Min. Metall.* no. 648, Nov. 1960), 49–69.

3. Ingham, F. T., and Bradford, E. F. The geology and mineral resources of the Kinta Valley, Perak. *Mem. geol. Surv. Malaya* 9, 1960, 81.

4. Reference 3, p. 80.

5. Alder, H. L., and Roessler, E. B. *Introduction to probability and statistics.* (San Francisco and London: W. H. Freeman and Co., 1960), 252p.

General Practices on Large-scale Hydraulic Mines in Malaysia

S. PAKIANATHAN, A.R.S.M., B.Sc., A.M.Aus.I.M.M., Student I.M.M.
Research Division, Department of Mines, Malaysia

P. SIMPSON, A.C.S.M., Student I.M.M.
Mining Engineer, Osborne & Chappel, Ipoh

SYNOPSIS

Current practices on large-scale hydraulic mines in Malaysia are described in general terms to give an overall picture of the various factors determining the applicability of the method and the stages comprising the operation without going into mechanical and technical details; no account is given of mines operating with earthmoving equipment.

The exploitation of the alluvial tin deposits of Malaysia by hydraulic methods has been carried out on a considerable scale for some 60–70 years, although for many centuries there has been small-scale surface mining by the indigenous people. Large-scale hydraulic mining practice has undergone a number of major changes over the years, culminating in the virtually universal adoption of a system of breaking ground with water jets and removing the spoil so produced by gravel pumps. Within the Malayan peninsula the most important individual tin-producing area is the Kinta Valley, which supports a large number of mines, both opencast and dredging.

Climatic conditions and topographical features of the area make available the supplies of surface water necessary for hydraulic mining, particularly on the eastern side of the valley where the steeply rising mountain range provides sources of water under natural head for mines operating in the valley below. This feature has accounted for the establishment of a number of large hydraulic mines in the Gopeng and Kampar districts and the practices described in this paper relate particularly to such mines.

GEOLOGY AND TOPOGRAPHY (Fig. 1)

The Kinta Valley lies approximately N–S and is about 25 miles long by 14 miles wide at its southern extremity. The valley floor is practically flat between the steeply rising mountains of the Main Range and the Kledang Range. The Kinta River and its tributaries drain the area,

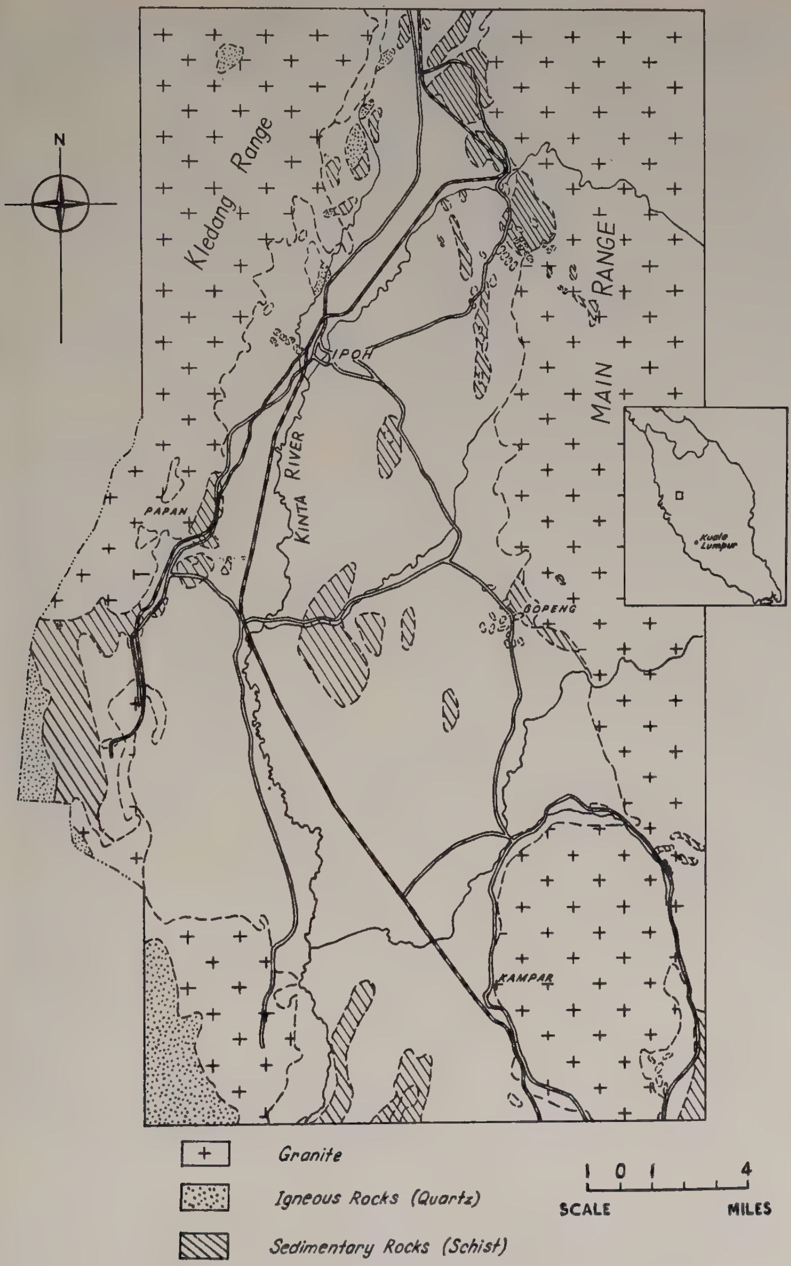


Fig. 1.—The Kinta district. (Geology reproduced by permission of the Director, Geological Survey of Malaya.)

the river system being generally, in view of the flatness of the valley, of low velocity. The mountain ranges are granite masses while the bedrock of the valley floor is generally limestone with occasional isolated occurrences of schist more especially on the granite–limestone contact, where this rock may form low foothills. Almost the entire valley floor is covered by a layer of alluvium, varying in thickness from a few feet to over 200 ft, which was formed mainly by the erosion and weathering of the granite masses. Both eluvial and alluvial deposits are found, the former being much less extensive and apparently of greater age. Prior to the deposition of the alluvium the limestone of the valley floor had been differentially eroded by groundwater forming in places a highly irregular surface of pinnacles and depressions which subsequently acted as natural riffles to concentrate heavy minerals from the weathered material as it was brought down by rivers and streams.

The nature of the alluvium varies considerably over the area, but it is generally a fine, relatively soft, sandy clay containing varying quantities of gravel and pebbles, and ranging in colour from white when much kaolin is present to reddish brown when iron minerals occur. The predominant heavy minerals found are ilmenite, cassiterite, monazite and zircon in various proportions, which are encountered sometimes in fairly clearly defined layers or beds, but more often as a general dissemination throughout the clay. The size range of the cassiterite is wide, covering from $+\frac{1}{4}$ in. to -300 mesh B.S.S. with the greater bulk being within the range 60–150 mesh.

The average grade of such heavy minerals is naturally extremely variable and, while it is probable that cassiterite is present throughout the entire alluvium, layers containing sufficient mineral to permit economic exploitation occur erratically. The economic cutoff or break-even grade is dependent on such factors as (*a*) metal price pertaining at the time; (*b*) depth of deposit; (*c*) nature of ground; and (*d*) nature of valuable minerals present.

The lowest grade of ground which could be profitably exploited at present under easy mining conditions (i.e. shallow sandy ground) is generally considered to be about 0·25 lb/yd^3. This is, of course, a minimum figure and under more normal conditions about 0·4 lb/yd^3 would probably represent an average limit of payability. Adverse mining conditions could further increase this figure, particularly should rehabilitation of the ground after mining be required, and under such conditions 0·6 lb/yd^3 might well represent marginal ground.

WATER SUPPLY

Rainfall in the region is abundant and relatively regular, averaging approximately 120 in. per year. Periods of heavier precipitation occur generally from October to December and again in March to April, while occasional droughts of up to a month are sometimes recorded. Exceptionally heavy rainfall, sometimes reaching 6 in. in 24 hours, occurs spasmodically and can be very troublesome to the opencast miner.

The steeply rising hills forming the eastern side of the valley provide sources of water at a sufficiently high elevation for use to be made of the hydrostatic pressure available when the water is conveyed by pipes to mines operating on the valley floor. In some instances it has been possible to utilize this water for motivating hydroelectric power stations while still retaining a sufficient elevation to give an adequate pressure below. Pipelines constructed for this service vary in diameter from about 18 to 45 in. and in length up to about 10 miles. Catchment and storage areas are generally of limited extent and capacity, thus giving rise to considerable fluctuations in the water supply, even a few days without rain being sufficient to reduce considerably the volume of water available at the mine.

Pressure in the pipelines is restricted to a predetermined figure, usually about 150 lb/in^2, by balancing input and output quantities. Such a pressure is sufficient for mining purposes without the necessity for very elaborate pipe systems; lower pressures reduce the effectiveness of the monitors, particularly in hard clays.

Distribution pipelines within the mining area range in size from 36- to 8-in diameter and are often of a slip-jointed type to facilitate installation and removal. Usually, one 8-in pipeline is provided for each monitor, which operates with $1\frac{1}{2}$- to $2\frac{1}{4}$-in diameter nozzles depending on the type of ground being treated.

Where it is impossible to obtain a water supply under natural head or when such supply is insufficient, engine- or motor-driven pumps are used to pressurize pond or stream water for mining—the system being either recirculatory or otherwise, depending on circumstances. These pumps are located as near to the mine as possible to reduce pipe friction losses, and pressures obtained are usually in the region of 75–85 lb/in^2; generally, a locally manufactured 10- by 12-in water pump driven by a 200-hp engine or motor is required to operate three 2-in monitors at 80 lb/in^2.

MINING

Hydraulic gravel-pump mining has a number of advantages over other alluvial mining methods.

(*a*) Topographical features of the area to be mined are relatively unimportant as undulating and hilly ground can be worked provided there is sufficient land for tailings disposal. The presence of standing water and/or swamps could, however, preclude such operations.

(*b*) Ground of varying depth can be worked with the same equipment, i.e. deep pockets in bedrock and local enrichments of limited area but considerable depth, as found in contact zones, can be exploited by this means.
(*c*) Complete extraction of values can be achieved as bedrock can usually be cleaned up.
(*d*) Selective mining can be practised enabling outputs to be balanced and unpayable ground to be left *in situ.*
(*e*) Working faces are visible at all times.
(*f*) Capital cost of equipment is lower than for other types of mining.

In principle, hydraulic gravel-pump mining is relatively simple and straightforward, consisting as it does of the use of high-pressure water jets to disintegrate ore-bearing ground, together with gravel pumps to elevate the spoil to a treatment plant. In practice, the operation is complicated by the need to maintain the water supply to a number of mine paddocks and, of equal importance, to remove this large volume of water combined with tailings sand and slime from the mining area. The maintenance of these services requires considerable forethought to ensure their non-interruption and non-interference with the development of the paddocks.

Initial mining operations consist of the establishment of the mine hole or paddock. This is achieved by sinking or cutting downwards with monitors and removing the spoil by pumping, the pump being lowered as the hole deepens. It is, of course, necessary to open out in area as the mine becomes progressively deeper in order to maintain a safe slope on the sides; this is generally about 1:1, but depends on the nature of the ground. When the paddock reaches the desired depth, either bedrock or some intermediate level in the case of deep deposits, the mine is developed by removing all ground above this level in the direction in which it is desired to mine. In this operation monitors are used to cut slots in the face, thus promoting controlled falls of ground. The fallen material is then disintegrated and washed down to the pumps. Monitors are advanced as mining proceeds—to keep them as close as possible to the face without exposing them or the operators to undue danger from uncontrolled falls of ground, it being clear that the closer the monitors are to the mine face being cut, the greater will be their effect.

As the mine face advances away from the pumps ditchlines are constructed in the floor of the paddock to conduct the slurry, produced by mining, to the pump sump. They are generally about 4 ft wide and graded about 1:40; by maintaining a high-velocity water flow the ditches minimize the redeposition of suspended solids in the paddock, which would occur to a marked degree if the slurry were permitted to flow to the pumps over a wide area and which would in turn lead to a build-up of material at the face, reducing the height available for cutting. When the paddock has been mined out to the desired limits of area the operation is repeated at progressively lower levels, should bedrock not have been reached. Provided that sufficient ground has been cut in the upper levels, the factor limiting the depth to which the mine may be excavated is that values encountered at depth must be sufficient to warrant the extra cost of multi-stage pumping.

Isolated patches of deeper ground or pockets in bedrock can be mined out individually by suspending small pumps from tripods or wire-rope slings and lowering them into the excavation, spoil from such operations being discharged into the ditchlines leading to the main pumps.

The gravel pumps currently being used are referred to as vertical gravel pumps, in which the motor frame of the prime mover and the pump casing are rigidly joined by a fabricated cage and the rotor and pump impeller are direct-coupled, the unit so formed being suspended during operation with its axis of rotation in a vertical plane. Prime movers are generally electric motors, though a few diesel-engined units have recently been introduced on small mines, and pumps are invariably of a single-stage centrifugal type. The general advantages of this type of pump unit compared with earlier 'horizontal' types are as follows.

(*a*) Being direct-drive there is no power loss in transmission from prime mover to pump.

(*b*) The unit requires no foundations but is simply suspended from a tripod or pump house piled into ground in the bottom of a paddock.

(*c*) By incorporating a chain block in the suspension the unit can be raised and lowered to allow for variations in water-level in the paddock and can be fully raised in the event of flooding.

(*d*) Units are quickly installed, being flexible and easily moved between sites.

(*e*) The pump operates on a negligible suction head and all power is therefore available for elevation.

The pumps most commonly used are those with 7- and 8-in inlet diameters driven by 100- and 150-hp electric motors respectively. Smaller units are available for working small areas of land or for mining out pockets in bedrock. Larger units are seldom used as their size and weight would restrict their application to more permanent sites and would tend to lessen the basic advantages already mentioned.

In selecting the size of pump for a particular application, it is usual to allow one 8-in pump to three 2-in monitors at 150 lb/in^2 or one 7-in pump to two monitors for lifts of 60–80 ft. If the head on the pump is not fixed, i.e. from bedrock to treatment plant, a maximum lift of between 70 and 80 ft is usually allowed when establishing an intermediate mining level. The size and speed of the motor has been fixed by standardization and it only remains necessary to select an impeller of the size which will give a maximum loading on the motor for the particular application. This is done in effect by trial and error as it is impossible to make full allowance for the pulp density and rate of flow of the slurry fed to the pump owing to variations in the ground being treated and fluctuations in water supply. It happens in practice that two or more similar units operating side by side in a single paddock will require impellers of different diameters to load the motors fully, due presumably to small differences in motor and pump characteristics and more especially to local variations in the pulp density of the slurry at the individual pump intake. Some examples of 8-in pump installations are given below:

Vertical lift (ft)	Motor hp	Impeller diameter (in.)	Current loading (max. amp)
49·6	150	21¾	200
66·4	150	23¼	200
69·7	140	22	180
77·0	150	23¾	200

It is difficult to be specific as to the performance of individual pumps in terms of slurry elevated in view of the number of factors involved, but an average for an 8-in pump operating on a 70-ft lift is approximately 3000 gal of slurry per minute at about 8–10 per cent solids.

Pumps currently used are manufactured from cast-iron in local foundries and have the advantage of being easily obtainable and relatively inexpensive; life of pump components is variable and depends mainly on the nature and quantity of solids handled. Pumps are connected to the upraise pipes by a flexible armoured rubber pipe, upraise pipes being of solid drawn steel 10 in. in diameter for 7-in pumps and 12 in. in diameter for 8-in pumps.

As an indication of the size of the paddocks referred to, a hypothetical 'average' operation could comprise a 30-in diameter water supply line providing for twelve monitors, say eight with 2-in nozzles and four with 1¾-in nozzles and requiring four 8-in pumps to elevate the spoil to the treatment plant; such an installation could be expected to treat about 150 000 yd^3 per month.

PRIMARY CONCENTRATION

The primary concentrator most commonly found on gravel-pump mines is still the *palong** or sluice box (although recently an increasing number of jig-cyclone plants have been introduced, particularly in the smaller mines). This much maligned piece of equipment has survived, essentially unaltered, over the ages because it is non-mechanical, easy to construct and simple to operate, it can handle large volumes of dilute pulp, and it can cope with widely fluctuating loads and large variations in pulp density of the feed.

---

* *Palong*, *takongs* and *dulang* washing denote local words for sluice-box, riffles and panning, respectively.

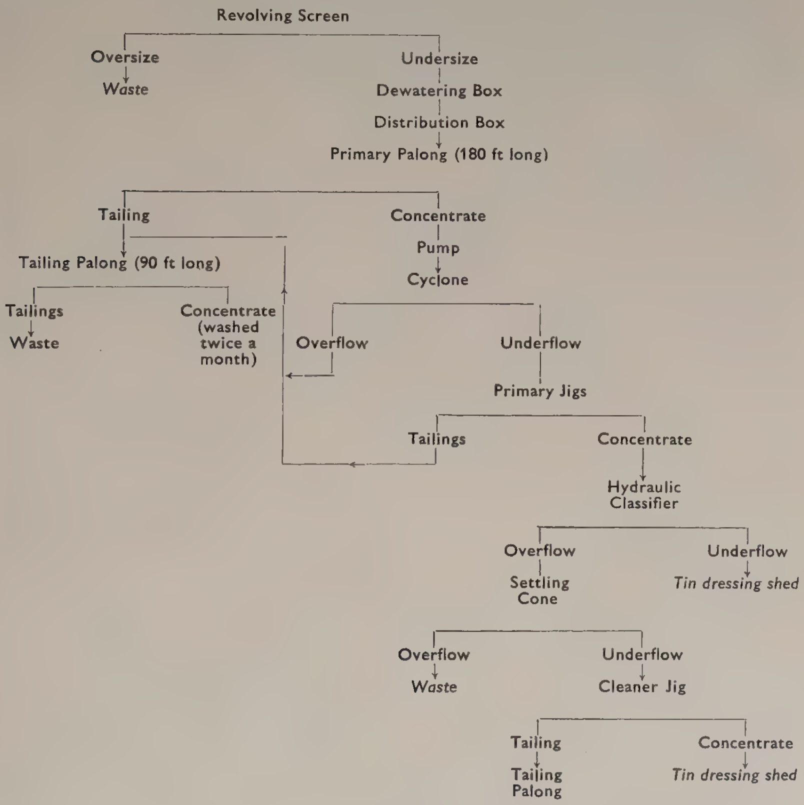


Fig. 2.—Flowsheet of treatment plant I (capacity 35 000 yd^3/month).

The principal disadvantage of the *palong* is its inability to concentrate successfully a feed containing large proportions of very fine valuable mineral, particularly in ground containing much slime. It is usually built of sawn timber and consists, in effect, of several individual longitudinal sections or sluice-boxes which are generally 4–6 ft wide and up to 270 ft long. The number of such boxes used is, of course, dependent on the volume and type of ground to be treated; in practice, one box is allowed per 100–150 cumins* of slurry, the lower figure being applicable to finer feeds. The inclination or 'slope' of the *palong* also depends on the nature of the feed, and is lowest for a fine-sized feed, the average being about 1 in 35. At the head of the *palong* there is an additional section which contains a revolving screen or a grizzly for the rejection of oversize material followed by a distributor or 'feeder *palong*' which enables the feed to any individual box to be controlled.

The sluice-boxes are divided laterally by removable riffles or *takongs* at about 16-ft intervals, their purpose being to create a series of pools where the velocity of flow is reduced, thus encouraging the deposition of solids. The riffles are increased in height as the *palong* fills and they are removed when the *palong* is to be cleaned.

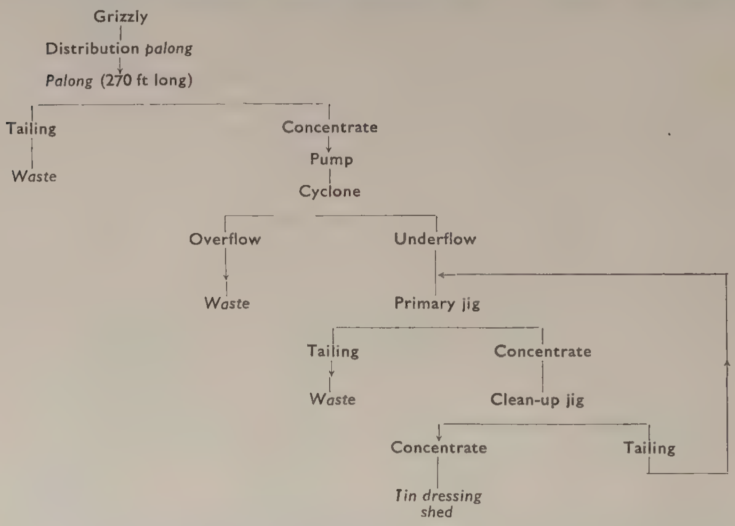


Fig. 3.—Flowsheet of treatment plant II (capacity 150 000 yd^3/month).

Until recently, the normal method of recovering the primary concentrate from the *palong* was by panning or *dulang* washing—the flow of slurry to the particular box being discontinued during cleaning—but this operation, in some circumstances, gave rise to losses of values particularly in the finer size range. Investigations carried out on the application of jigs to the clean-up of *palongs* have evolved simple jig plants operating on cycloned feeds, which to date have been successful and have indicated considerable improvement in the recovery of fine, −150-mesh cassiterite. Two alternative flowsheets (Figs. 2 and 3) have proved successful.

---

* Cumin = 1 ft^3/min.

TAILINGS DISPOSAL

The formation of large volumes of dilute tailings slurry is inherent in hydraulic mining and the disposal of such material presents considerable problems.

Wherever possible, a system of dumping back is adopted whereby tailings from current operations are dumped into previously excavated mine holes. This effectively disposes of a large proportion of the sand tailings, but does not account for the swell in broken ground if the new paddock is as large as the one being filled, nor will it normally retain more than a small percentage of the contained slimes. Additional sand may, of course, be retained on the area by stacking behind dams using pumps to elevate a high pulp density slurry from the tailings ditchline. This method is often used, but naturally increases the overall cost of mining. The remaining slime effluents from the various paddocks on a property are often then conducted by ditchlines to extensive central slimes settling areas where the remaining suspended solids are settled out. Owing to the sluggish nature of most of the rivers in this region, which consequently restricts their carrying power, the solids content of mine effluents ultimately discharged into rivers is severely restricted and controlled by mining laws. Large areas of land, extensive dams and systems of ditchlines are necessary to achieve the virtually clear final effluent required.

One aspect of this matter which is becoming increasingly important is the use of tailings to effect a reclamation of land currently or previously mined, particularly adjacent to towns and villages which are now developing and expanding, and often mining leases in such areas are only issued or renewed conditional upon the lessee agreeing to leave the land filled, levelled and drained on completion of mining operations. It is obvious that such an operation could add greatly to the cost of mining and it is therefore necessary to select a means of effecting the reclamation at the lowest cost.

In the case of large projects of this nature the provision of suitable earthmoving equipment may be warranted. However, in a majority of cases the volume of ground involved is insufficient to justify the capital outlay which would be necessary, and gravel-pumping methods have been adopted. To minimize operating costs it is necessary to increase the pulp density of the slurry compared with that normally pumped in mining operations.

This can be achieved by recirculating a proportion of the slime fraction and using this in place of clear water for the monitors. The greater pulp density of this mixture compared with water gives rise to an increase in carrying capacity and a greater quantity of sand pumped will result. Pulp densities of slurry pumped to filling areas of up to 15 per cent can be obtained, compared with the more usual 8–10 per cent normal in mining. As heads encountered are usually low, large-diameter pump impellers are employed and oversize delivery pipes are used to reduce frictional losses. The pump intake is partially choked in the sand and excess water from the sump is diverted elsewhere.

Acknowledgements.—The authors wish to thank Messrs. Osborne and Chappel, Ipoh, and Mr. C. B. Du Vaure of Société Anonyme des Etains de Kinta for providing, and permitting the publication of, information contained in this paper.

622.271.1:551.46

Paper 7

Mining Offshore Alluvials

M. J. CRUICKSHANK, A.C.S.M., M.Sc., F.G.S., A.M.I.M.M.
Institute of Marine Resources, University of California; at present Assistant Professor, Mining Engineering, University of Alaska

---

SYNOPSIS

Potential mineral reserves of nearly all the major metals and non-metals, as well as many of the less important elements, are contained in the ocean and on and under its floor. Exploration methods, with one or two exceptions, follow the same principles as conventional exploration for blind orebodies on land. Such aids as low-frequency reflection surveys, television and bottom photography are peculiar to the undersea environment. Others, such as geochemical sampling, and photogeology, have not yet been applied, or are used only very close to shore. Alluvial deposits offshore are mined by dredging, using bucket ladders, draglines, clamshells, hydraulic dredges, and airlifts. Accurate position location for control of both sampling and mining is still a major problem, and others are associated with mining control, waste disposal, and legislation. Current operations (1963) for precious metals, tin, diamonds, iron ore, sand and gravel, and lime sand are described, as well as developments in the United States for mining manganese and phosphorite nodules to depths of 4000 ft.

OCEAN MINERALS

Lying beneath the waves of the oceans and on and under their floor is an enormous wealth of potentially economic minerals. Sea water contains, in each of its 330 million cubic miles, over 150 million tons of dissolved salts, including appreciable quantities of lead, zinc, copper, uranium, silver and gold, and carries trace amounts of almost every other known element. The gold content alone has been valued at from $25 thousand to $25 million per cubic mile. Salt, magnesium and bromine are the only chemicals extracted commercially from sea water at the present time, but a breakthrough in extractive chemistry should prove many other minerals to be economic.

The Ocean Floor

On the deep ocean floor, mineral concentrations occur in a great variety of forms. The best known of these are the much publicized manganese and phosphorite nodules which overlie vast areas of the Pacific and Atlantic Oceans and their borders. Deep-sea manganese deposits are formed by ionic precipitation of colloidal manganese and iron oxide on any hard surface on the sea floor. Other metals are scavenged by the iron and manganese during transport from their point of origin. The most prolific form of these deposits is as nodular grains or pieces up to several feet in diameter lying on the surface of the sea floor at depths of from 3000 to 19 000 ft. An average composition of manganese nodules from the Pacific[1] is given as manganese 32·4 per cent, iron 18·5 per cent, copper 0·80 per cent, nickel 1·14 per cent, and cobalt 0·47 per cent, although the proportions of these metals vary widely in different locations. The same source gives a nodule density of around 11 000 tons per square mile over an area of millions of square miles. The phosphorite nodules, which contain as much as 30 per cent tri-calcium phosphate, are formed in a similar fashion to the manganese nodules, but at lesser depths. These have been located off the west and east coasts of the United States, Mexico, Argentina, Peru, Chile, Spain, South Africa, Japan, and in the Indian Ocean, at depths varying from 400 to 8000 ft.

Also in the deep sea are countless billions of tons of clays, muds, and oozes, carrying relatively high concentrations of copper, aluminium, cobalt, nickel, lime, silica, phosphorus, barite, tin, rare earths, vanadium and zirconium.

The Continental Shelf

Economic mineral deposits similar to those occurrences found on land also occur on and within the continental shelves. The continental shelf is that part of the continental land masses which lies submerged beneath the sea between the tide line and (for simplicity) the 100-fathom line. At this depth the shelf terminates in the steeply dipping continental slope leading to the deep sea. Generally described as flat, the continental shelf is in fact of very varied topography and structure. The depth of its edge ranges from a few fathoms to over 600 ft and averages 432 ft, its width varies from zero to over 350 miles, and its total area is around 10 million square miles or 15–20 per cent of the total land surface. The fact that this area is at present submerged is entirely fortuitous, and geologically the present location of the edge of the sea is only temporary. Being a part of the 'granitic' continental structure, the submerged shelves are geologically similar to the adjacent land masses and have a mineral potential of the same order. In the Americas alone the shelf amounts to some 3 700 000 square miles. This is just slightly less than the total area of Canada and is still virtually unexplored.

Apart from the fluid overburden there is little distinction between the type of consolidated deposit found on the shelf and on land. Many offshore deposits at present in production are extensions of orebodies outcropping or worked on land, such as bedded coal and iron deposits, but all known types of hard rock deposit may be expected. Outcropping deposits will show different effects of alteration due to the sea water, and there may be other products of sea-water leaching that are not yet apparent, but the conventional types of deposit should predominate.

Although offshore operations for petroleum and natural gas are extensive, and underground mining operations are carried out offshore for coal, iron, sulphur and tin, the majority of deposits now being worked are unconsolidated and may be termed alluvials. To date, all these operations are carried out on the continental shelf. Nevertheless, plans are currently being made for mining at depths of around 4000 ft. The improvement of operating techniques and the knowledge gained in the mining of alluvial deposits on the continental shelf will serve as a preliminary to the advance into deeper waters.

Current alluvial mining operations offshore include the search for submarine placer gold off Nome beach in Alaska, the extraction of diamonds in South West Africa, iron ore in Japan, lime sand in Iceland, and oyster shells from San Francisco Bay. Some of these operations are described in greater detail later in the paper.

EXPLORATION METHODS

With one or two exceptions, mineral exploration offshore follows the same principles and uses the same techniques as conventional exploration for blind orebodies on land. Extrapolation of the known geology of the adjacent land areas is a prerequisite to the study of any offshore area. Although many of the usual techniques of search for blind orebodies are practised, such things as low-frequency reflection surveys, television or close photographic bottom coverage and so on, are peculiar to the marine or offshore environment. Others, such as photogeology or geochemical sampling, do not apply or have only been used very close to shore. In exploration for unconsolidated deposits offshore, seismic and gravimetric surveys are not normally employed. Initial bottom surveys are carried out in the favourable area using the gas-exploder or sparker, a low-frequency (150–300 c/s) echo-sounder which not only records the bottom profile but penetrates the sub-bottom down to depths of 1500 ft. Distinction between sediment and bedrock, and even sediment type, can clearly be made from the continuous record put out by this instrument (Fig. 1). Traverse lines are run at regular intervals to give systematic areal coverage. At this stage the necessary physical features, such as stratigraphic traps, submerged beaches, or drowned river valleys, may be delineated or it may be desirable to define further the areas of potential ore by additional methods. The magnetometer has been used successfully to locate concentrations of magnetiferous sands on submerged beaches, a good guide to placer gold; and bottom photography, supplemented by television has been used to delineate and evaluate areas of marine nodules. These practices precede actual physical sampling. Geological sampling may be carried

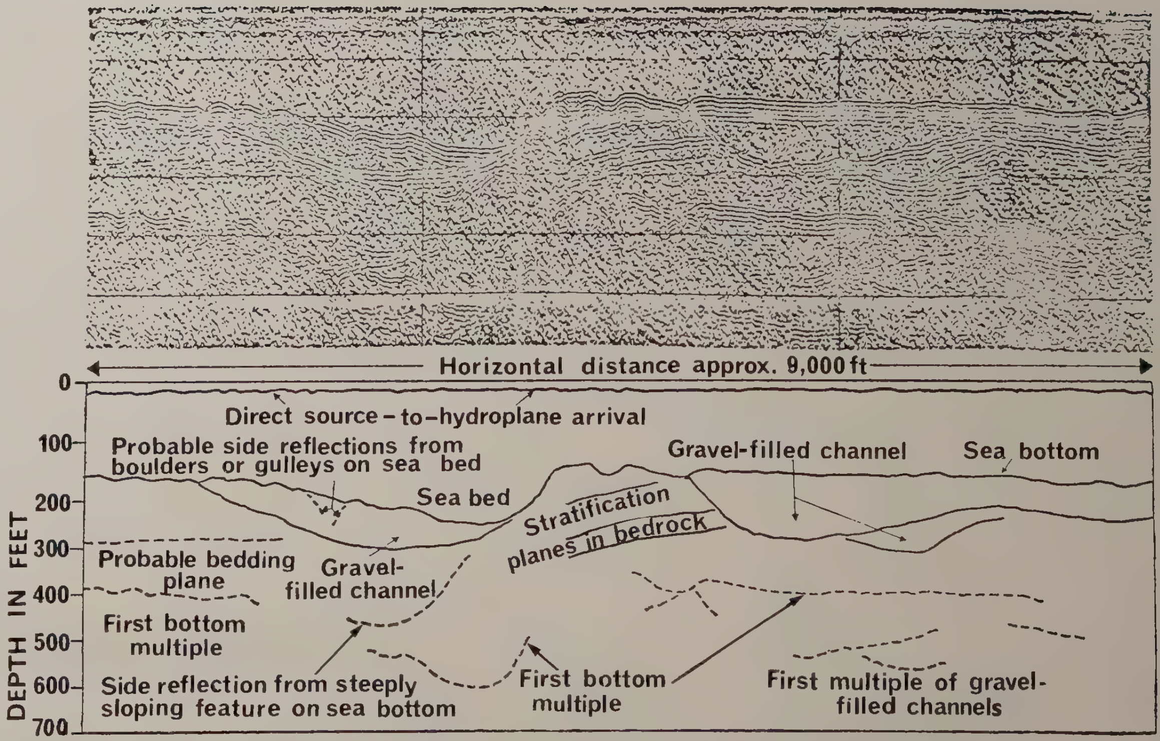

Horizontal distance approx. 9,000 ft
Direct source-to-hydroplane arrival
Probable side reflections from boulders or gulleys on sea bed
Sea bed
Gravel-filled channel
Sea bottom
Stratification planes in bedrock
Probable bedding plane
Gravel-filled channel
First bottom multiple
Side reflection from steeply sloping feature on sea bottom
First bottom multiple
First multiple of gravel-filled channels
DEPTH IN FEET
0
100
200
300
400
500
600
700

Fig. 1.—Sparker record from Channel tunnel survey, with explanatory key.[2]

out by dredging, drop coring, or drilling, and by divers or using submarine vehicles. Where a general sample of the bottom is required to differentiate bottom sediments or delineate sand-bearing areas, a grab sample may be sufficient, but to evaluate deposits in depth a drilling method is necessary.

The drill most commonly used in evaluating alluvial deposits offshore is a modification of the churn drill operated from a barge or floating platform or, particularly in South East Asia, adapted for hand power entirely. The drills are used in water depths up to 50 ft. Very careful supervision is required to minimize sampling errors caused by bailing ahead of the casing and careless handling of the sludge. Where this drill is used consistently in the same type of ground a grade factor may often be developed and applied successfully to eliminate bias. More recent developments have been in the use of airlifts, suction pumps or venturis for sludge removal in combination with large-diameter (24-in) casing and a rotary drill stem. Drop coring is effective where the sediments to be sampled are soft enough to be penetrated by the free falling tube. Examination and sampling of the bottom by scuba divers is now commonplace, while small submarines designed for visual inspection and sampling below the depth range of free divers are now on the market and have been used in the evaluation of deposits of nodules.

During drilling, the vessel is normally held in position by four to six anchors splayed out from it; but spuds, or stakes, driven into the bottom may be used if the water is very shallow.

For drilling in deep water, a method known as dynamic positioning has been developed. A series of outboard engines mounted on opposite sides of the vessel give it extreme manoeuvrability. Position is maintained by automatic centering in a circle of sonar reflectors placed around the drilling target, either on the bottom or suspended by taut wire buoys. Several drilling ships are now equipped with this facility. In the completion of the initial Mohole deep-water drilling project it was possible to maintain the ship within 100 ft of the vertical projection of the drill-hole collar over 2 miles below.

POSITION LOCATION OFFSHORE

One of the major problems in operating offshore is accurate positioning for control of both sampling and mining. Most operations at the present time being within sight of land, it is possible with visual survey by triangulation from shore stations to maintain an accuracy of ±6 ft. During the sampling or traversing, radio contact with the boat is maintained by the shore tracking stations and the exact location at the time of sampling, or at any point during the traverse, is recorded. In preliminary investigations the sextant-chart technique is often used, sighting known landmarks on shore from the boat, with an accuracy of ±50 ft. Where free divers are employed, the relative position of the diver from the diving boat is determined by using marker buoys, lines or the position of the diver's bubbles, account being taken of local current or wind conditions. Small sonar guides are used by divers to locate themselves relative to the boat or other object. This equipment is not yet perfected to read off distances, however.

Out of sight of land, standard navigational aids must be used. Apart from the normal star and sun sights, there are several types of electronic position locators, such as Loran, Shoran and E.P.I. Star and sun sights will give accuracies in the region of ± half a mile. Loran, or long-range navigation, is based on the difference of arrival time of two radio signals and has a similar accuracy. Shoran, or short-range navigation, requires two shore stations within visual range of the boat, and is accurate within a few feet. E.P.I., or Electronic Position Indicator, utilizes a pulse-type arcuate system using low-frequency radio waves to measure the time taken by a pulse transmitted to a shore station to be returned to the ship. Two shore stations are necessary and an accuracy of ±200 ft is obtainable at a distance of 275 nautical miles. Transit, recently developed by the U.S. Navy, uses artificial satellites and will give a similar accuracy at any point on the globe. Once an area is accurately located by at least one point, local control survey grids can be laid out by means of buoys and bottom sensors which will give control as accurately as on land.

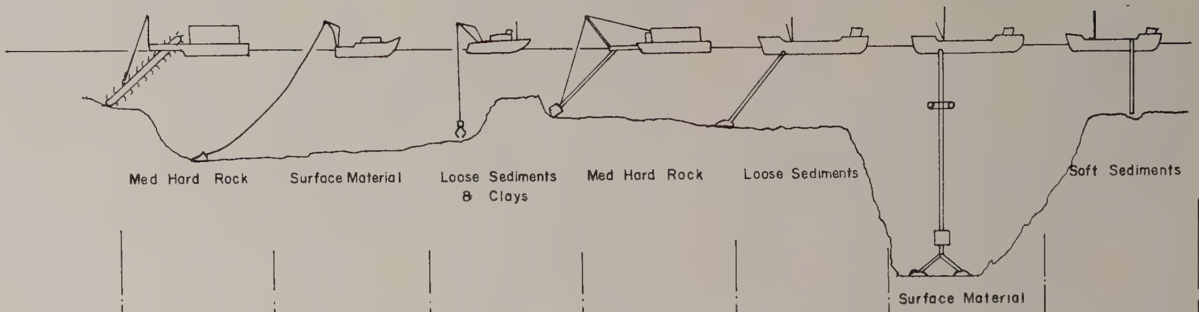
MINING

Alluvial deposits offshore are mined by dredging using bucket ladders, draglines, clamshells, hydraulic dredges and airlifts (Fig. 2). None of these systems is new in concept and not all have been successfully used in mining operations, but each is applicable to particular cases.

The mining dredge is a completely self-contained mining and beneficiation plant, built on to a floating hull. The unit processes performed by the dredge are excavation of the overburden and the ore, beneficiation of the ore to produce a concentrate and disposal of the overburden and waste material. Mineral concentration and waste disposal are independent of the method of excavation. Upon the choice of excavating equipment the success of the mining operation primarily depends.

Bucket-ladder Dredges

Mining with the bucket ladder is a continuous process of excavation using an endless chain of steel buckets which dig into the ground below the level of the floating dredge (Fig. 3). It is quite a common civil engineering technique and possibly among the first of the automatic production lines. Bucket capacities of mining dredges vary from 2 to 20 ft^3, giving monthly capacities in the range of 60 000–500 000 yd^3. Digging is



	BUCKET LADDER	DRAGLINE	GRAB BUCKET OR CLAMSHELL	HYDRAULIC WITH CUTTER HEAD	SUCTION OR HOPPER DREDGE	HYDRAULIC DEEP SEA PROPOSED	AIRLIFT
Working Depth – Current	0' - 150'	0' - 50'	0' - 200'	0' - 100'	0'- 150'	——	0' - 100'
Working Depth - Proposed	250'	5000'	800'	250'	——	15,000'	600'
Capacity yds./h	300 - 3000	1 - 400	60 - 15,000	250 - 4500	400 - 6000	230 - 1000	50 - 500

Fig. 2.—Various dredge types used in the mining of alluvial deposits offshore.[3]

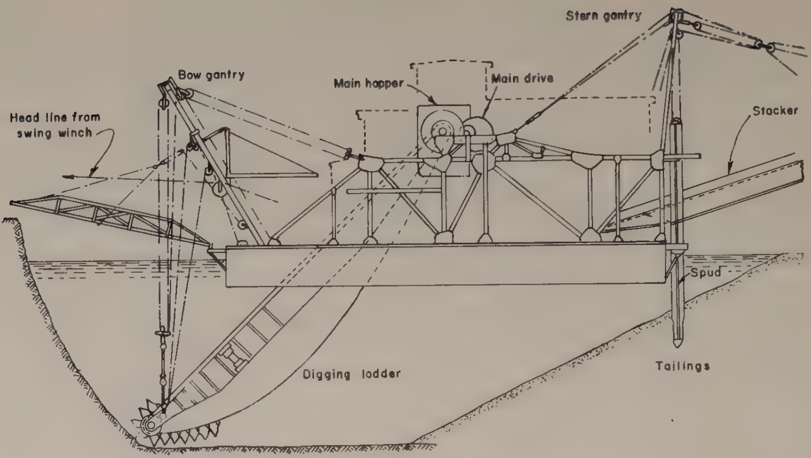


Fig. 3.—Details of a California-type bucket-ladder dredge.[4]

normally to 100 ft below waterline and, on occasion, 150 ft. Concentration of the mineral values is carried out on board the dredging vessel and tailings are discharged to the rear. Since most mining operations are for high-gravity minerals of low volumetric concentration, the tailings constitute the major part of the plant throughput.

Bucket-ladder dredges are of two types—the spud or California type, and the headline. In the California type, large spuds, or vertical stakes, are dropped from the stern of the dredge which is pivoted alternately on each one of these during the cutting sweeps, advancing forward with a kind of walking motion. The spuds normally bottom in the tailings which are discharged closely behind the dredge. This type of dredge is used mostly in the mining of river or land placer deposits, with the dredge floating in a water pond little larger than the dredging unit itself. Where the deposits are completely submerged, as in swamps, estuaries or offshore, the headline dredge is used. The headline is attached to an anchor two to three thousand feet ahead of the vessel and additional stabilizing anchors are put out from each of its corners. The anchors are placed by boom or by small craft that are kept with the dredge. In the presence of tides or moving waters, the dredge is able to ride on the lines which allow a vertical movement not available in the California dredge.

Dredge capacities depend on the bucket size, the horsepower and the digging conditions, but normally average from about 150 000 to 400 000 yd^3/month. Generally, the depth of the excavation has little influence on the capacity if it is within the designed limits. Major factors influencing the digging conditions are the presence of trees, stumps, boulders, and so on, and the hardness of the bedrock underlying the pay ground. Since most heavy minerals, particularly gold, will concentrate on, or close to bedrock, excavation of the top layer of this material may be necessary and may cause considerable wear and tear on the buckets. Bucket damage may also occur when working to bedrock during rough weather offshore, owing to the impingement of the ladder on the bottom, and while hydraulic balances eliminate this shock loading, it is usual to stop work and either ride out the weather on the headline, or if working with spuds, go to shelter.

Power on the dredge is required to actuate the bucket line, control the winches, and run the pumps and other plant, such as trommels, jigs, and concentrating tables. Electric power is often supplied by cable from shore, but in out-of-the-way places diesel-electric plants and even steam generators are used. Steam has certain advantages in primitive areas where there is wood fuel and the strength and simplicity of the machinery allows the use of unskilled mechanics, but it is inefficient and not recommended generally. Recent advances in dredge power plant design have been the use of Ward-Leonard controls, eddy current couplings and silicon rectifiers. The Ward-Leonard control is familiar to mining men in winding engines and allows automatic adjustment of power on change of bucket load, thus permitting a higher bucket speed to be maintained without the danger of overloading the bucket line. Normal electrical supply from shore is 11 000 V a.c., rectified on board the dredge.

Dredges normally work a 24-hour shift and for the size of the plant require very few men, the crew often consisting of a dredgemaster, winchman, two oilers and a millman—about five men per shift.

Draglines and Clamshells

Because of its simplicity the dragline is often used in the sampling of deep-sea deposits at depths as great as 30 000 ft, but there is no record of its having been used for mining offshore except in the recovery of beach sands above the high-water mark. Recent studies[13] indicate that as a method of commercial dredging for nodules in deep water it might be applicable to a depth of 5000 ft, but would not be efficient in cleaning the bottom. Very large and unwieldy buckets would be necessary for such an operation and control of the dredge sweeps would be difficult.

Clamshells, or grab dredges, being suspended from wire ropes are similar in concept, but are becoming more widely thought of as a mining tool. Grabs were first introduced in 1865 and later the multi-line grab was developed, its great advantage being that it could be opened underwater if caught on an obstruction. Since then, many general advances have been made in the design and digging efficiency of grabs, and today the four-rope heavy excavating type can be regarded as one of the foremost digging tools. From a tonnage viewpoint it probably handles more material than any other single mechanism. Outputs of 800–1000 t/h are common with a single grab, and 30-t capacity grabs, with outputs of 1500 t/h, are already beyond the drawing-board stage.

Certain requirements have been suggested for the design of grab dredges for offshore mineral working,[6] namely:

(1) The action of the grab should be continuous: it should be brought from its digging to its dumping position in one motion without secondary motions such as slewing and luffing.
(2) The dumping hopper should be in a position to allow direct feeding of the plant without the use of conveyors, belts, etc.
(3) The grabs should be able to traverse a wide area of ground from a stationary pontoon without constant winching.
(4) The grab should be held stationary during dumping as waiting for swing abatement at sea can be particularly time-consuming.
(5) The mechanism should incorporate the means of counteracting swell in offshore dredging, avoiding the danger of the grab being plucked out of the ground during the closing cycle. This may be done by a compensator which also has the feature of retaining the grab in an upright position on uneven ground before closing.
(6) The mechanism should be completely and fully automatic in all its functions over the entire grabbing cycle, whatever the depth, within the limits of the winding rope. This would eliminate operator guesswork and not only ensure full loads, but enable the bottom to be cleaned up systematically and to a greater extent than is possible by any other means.

In the past, grabs have been designed for mining at depths of 800 ft, utilizing the differential hydrostatic head to close the bucket jaws. Others have been fitted to converted oil tankers, but have been only partially successful. More recently, both in Japan and in Great Britain, grab dredges for mining are being designed and built to include all the requirements listed above. A typical dredge now being constructed is the Dinosaur, designed for the mining of alluvial tin in South East Asia (Fig. 4). Some notable features of this dredge are twin 6-yd^3 buckets which will give a monthly output of 308 000 yd^3 in 135 ft of water, at a power cost of 1·4 d./yd^3 dredged. To enable an accurate dredging pattern to be followed, the crane structure is mounted on bogies running on a circular track. After each operation of the grab, the crane is indexed round through approximately 12°, thus positioning the grab automatically for the next dip. A concentration plant has been designed specifically for the dredge to give it a high recovery, particularly of fine tin. An additional power cost of 3·5 d./yd^3 is estimated for the recovery plant. Apart from the low initial cost, the advantages claimed

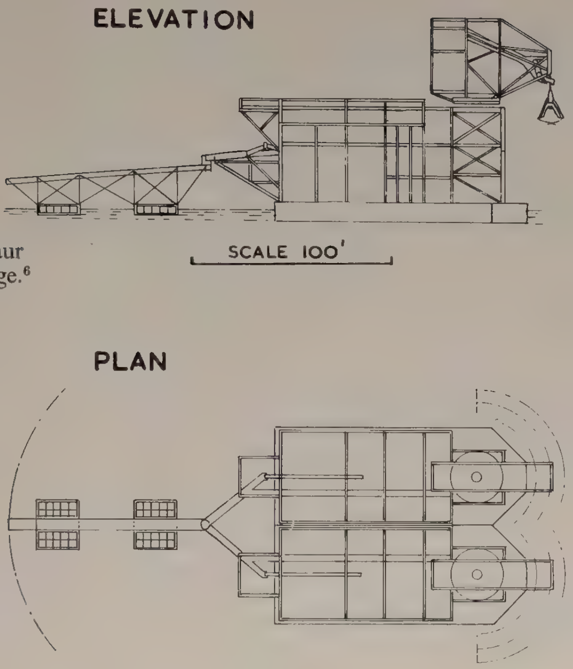


Fig. 4.—The Dinosaur grab bucket dredge.[6]

for this unit include a low weight compared with the bucket dredge, the all-up weight being just under 2000 tons; the ability to work to any depth and remain stationary for comparatively long periods before it is necessary to move to cut a new bench; low maintenance costs; the ability to turn and work right angles at the end of a cut; good stability; the ability to clean up the bottom systematically and winkle out enrichments from the interstices between limestone pinnacles; and the fact that the right kind of grab for any particular type of ground is available and can be quickly fitted with very short downtime. As an offshore dredge the clamshell or grab dredge can be worked beyond the limits of a bucket-ladder dredge and in a considerable swell without loss of output.

Hydraulic Dredges

The use of hydraulic dredges in mining is confined mainly to the removal of overburden or the mining of unconsolidated sand and gravel deposits. Low[7] has stated that several attempts have been made to apply suction dredging to mining, but so far none is known to have been successful. However, recent improvement in size and capacity of suction dredges for civil engineering has been substantial, and the increase in mining operations offshore and proposals to 'vacuum-clean' the deep-sea floor for nodules point to an inevitable revival of interest in the hydraulic dredge as a mining tool.

One of the fundamental problems in suction dredging is the continuous entry of the maximum amount of solids into the suction pipe. Hydraulic (or suction) dredges may be built with or without cutter heads, those with cutter heads being used to dig into consolidated alluvium or soft to medium-hard rock, and those without to remove unconsolidated sands, muds and silts (Fig. 5). Most mining operations use cutter heads, normally of the rotating, hollow-bit type, placed directly in front of the suction pipe, to break up the ground and direct the flow of solids into the pipe. The pipe and rotating shaft are attached to a rigid ladder which is raised or lowered in front of the dredging vessel. Dredges are now built with a rotating suction pipe which has the cutter head directly attached. The vessel is generally anchored at the stern by hawsers, or by spuds driven into the bottom, and is swung in an arc from the point of anchorage, to give a maximum sweep to the cutting head.

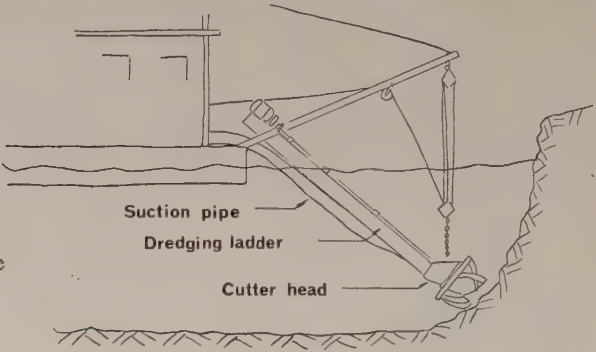


Fig. 5.—Suction dredge with cutter head.

Without the cutter head, there are various methods for gathering the material into the suction pipe, including the use of a plain suction head, the dust-pan type, or the drag head, the suction head being held stationary, thrust forward, or dragged. Water jets to agitate the solids are often used with the plain suction head (Fig. 6), and a ladder mounting for the suction pipe of the dust-pan dredge may be incorporated, but drag heads are generally attached to a rigid suction pipe which swings down from the side of the vessel and is suspended at the head. Joints are also used where extra pipe flexibility is desirable. Small suction dredges utilizing the jet-venturi principle have successfully operated in the mining of gold by skin divers from river bottoms in California and Colorado (Fig. 7).

Suction dredges generally operate at depths of 30–90 ft, but are being built for depths up to 250 ft. Preliminary designs have been made for

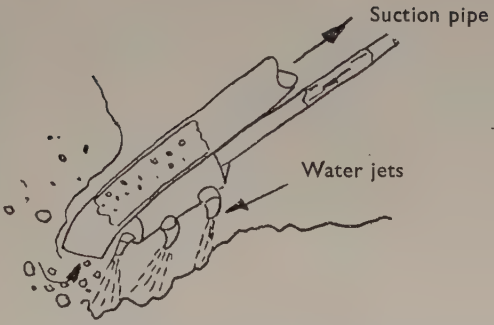


Fig. 6.—Suction dredge with water jetting.

hydraulic dredges to mine the deep oceans at depths of 4000 ft and greater, but none of these has yet been built. Velocity head and entrance loss to the suction pipe are independent of dredging depth, but at great depths the increase in the friction head and in the head required to support and move the column of mixture will obviously require submerging of the pumping unit.

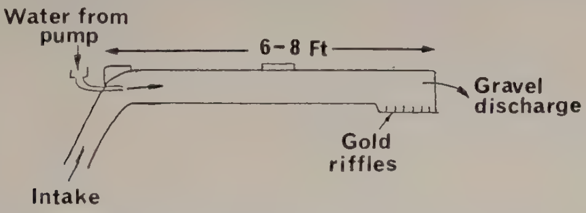


Fig. 7.—Venturi suction dredge for gold, as used by divers.

Hydraulic dredges generally have a high capacity: a 10-in dredge with 400 hp on the pump and a 75-hp cutter will conservatively pump 250 yd^3/h in soft material and 50 yd^3 in soft rock; a 30-in dredge with 8000 hp on the pump and 2500 hp on the cutter will pump 4500 yd^3/h in soft materials and up to 2000 yd^3 in soft to medium-hard rock.

Airlifts

The airlift (Fig. 8) works on the principle that air injected into the bottom of a pipe, submerged about 60 per cent of its length in water will produce a density differential in the pipe. The air–water mixture in the pipe will then flow upwards under the influence of the hydrostatic head and this flow will cause a suction at the bottom end of the pipe which can be very substantial. Airlifts have been used in the past for numerous civil engineering projects where simplicity of construction is important as they consist of only two pipes; they may be made in any

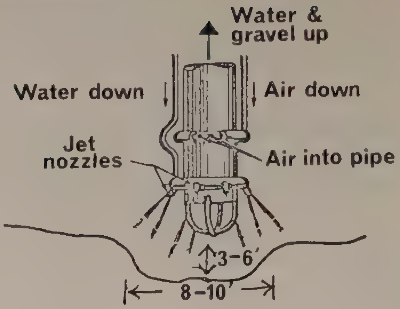


Fig. 8.—Detail of airlift head (with water jetting to stir up sediment).

field shop and require only compressed air to operate. Such a field-built dredge was used by the U.S. Navy to dredge silt from the harbour in Subic Bay at a depth of 57 ft. A 2-in pipe delivered compressed air at 75 lb/in^2 into a 20-in rise pipe with a bell-shaped mouth. Held stationary on the bottom, the dredge scooped a 10-ft diameter hole from 3 to 5 ft deep in the silt. The same dredge was used in depths of 70 ft of water and with lifts up to 40 ft above the surface. The principle of the airlift is by no means new and it has been used in the oil industry for many years for pumping deep wells and excavating trenches for undersea pipelines. It was undoubtedly the familiarity with its capabilities in this field that led the Collins group to select the airlift as the mining dredge for their South West African diamond operations. Since there is no suction line it is possible to use a rubber tube for the dredge pipe, which allows great ease of handling, particularly in rough weather. Further development of this type of dredge is almost certain, and its use has been proposed for very deep water.

Waste Disposal

Waste disposal and mining control sometimes present a joint problem offshore. In many instances the waste from mining is dumped overboard, close to the dredging vessel, where it is liable to contaminate or dilute unmined ground. Sometimes it is possible to discharge the waste material out of the mining area through pipelines laid on pontoons, or on the sea floor, or even utilize it as fill for beach reclamation or land fill. But where the deposits being mined are extensive, far from shore or at great depth, the problem becomes more acute and has not yet been solved for all cases.

LEGISLATION

As more deposits of offshore minerals are discovered, State, Federal and International law will become more exact. Present laws still leave many unanswered questions which may not be resolved until specific cases come under dispute.

Within the three-mile limit, state laws and sovereignty apply. The limit, which in the Gulf States is 3 nautical leagues, each of 3 miles is measured from the adjacent coastline; but even here there are disputes, and California claims that her coastline extends seaward to include its outer islands.[8] New laws are slow in appearing, but Alaska has recently formulated mining laws relating to offshore operations.[9] These state that submerged lands (low tide line to the three mile limit) are open to leasing only. Offshore prospecting permits are provided for, and they give the holder a right to a mining lease if he finds workable mineral deposits. An offshore prospecting permit may cover a maximum of 5120 acres in a reasonably compact form (length not more than four times the width), and is granted for two years with a single two-year extension provided for if warranted. Submerged lands containing known deposits may be offered for lease by competitive bidding. Rental for offshore leases of $1.00 per acre per year, and expenditures on or for the benefit of the leasehold are creditable against the rental. No royalty is required. Canada has also composed some new laws to cover offshore lands but most countries rely on the adaptation of existing laws to cover any eventualities.

The rules that apply to the outer continental shelf of the United States, beyond the states' jurisdiction, were set up originally with offshore petroleum and sulphur operations in mind, and are contained in Public Law 212, Outer Continental Shelf Lands Act, passed 1953. Mineral exploration in this area requires initially an application to the Department of the Interior for a geological and geophysical permit. If exploration is favourable and it is desired to proceed further, the Bureau of Land Management, on request, puts the area up for bid. The initial discoverer does not have any priority on bidding and notice of offering the submerged lands for competitive bid lease is posted in the Federal Register at least 30 days prior to the offering. Bids are in terms of cash bonuses, lease, and/or royalty, and the leases are for specific minerals only.

International law, according to the Geneva Convention of 1958, gives a country sovereignty over its continental shelf in terms of benthic sea life and minerals. The shelf is defined as limited (*a*) to the sea bed and subsoil of the submarine areas adjacent to the coast but outside the area of the territorial sea, to a depth of 200 metres, or beyond that limit, to where the depth of the superjacent waters admits the exploitation of the said areas; and (*b*) to the sea bed and subsoil of similar areas adjacent to the coasts of islands. In effect, this says, that the shelf limits occur only beyond that depth at which it is possible to mine, which in a few years may well mean that there will be no limit. The United States ratified the convention in 1960, but to date too few countries have approved it to make it legally effective, and it is unlikely that with its present wording it can ever be so.

In the deep oceans the international law of the High Seas still applies, and the mineral resources are apparently the property of 'he who first reduces them to his possession'.

CURRENT OPERATIONS OFFSHORE

Some offshore mining operations have been going on quietly for a number of years, while others not yet begun have been given publicity. There is no doubt, however, that in moving offshore the mining industry is opening up a new, vast, and profitable source of minerals. Some of the current operations are described below.

Precious Metals

The Alaskan Division of Mines and Minerals received 46 new applications for offshore mining leases in April 1962; 27 permits were granted in the Nome area, one each at Teller and Tin City, four at Platinum, two at Kodiak, and five at Juneau.

Shell Oil Co. applied for prospecting permits on 5120 acres in Norton Sound, off Nome, and at least eight other individuals filed claims on an additional 27 200 acres in the same area.

Williams Hydraulics Inc., of Oakland, California, plans to prospect and dredge for gold in Stephens Passage, south of Grand Island, Alaska. The area covers 8 square miles, with water depths of 520–900 ft. John C. Williams, President of the Company, is reported to have invested $1 000 000 in the project since preliminary sampling was carried out in 1933.

The Shell operation at Nome

The beach at Nome, Alaska, is a narrow strip about 200 ft wide from which over $2 000 000 in fine gold has been washed. Other placer deposits, now mostly worked out, were found in old beaches of varying age, some original, some reworked. The richest was the furthest inland and the youngest of the raised beaches, which age from mid-Miocene through Pleistocene. The source of the gold is in the schists of the Anvil Creek drainage basin, where it is finely disseminated in the rock. Erosion processes separate the gold and it is carried to sea and concentrated by wave actions. The present beach is gently sloping and the water depth at the extent of the Shell lease is only about 60 ft.

Shell prepared composite geologic maps of the hinterland using existing data, and in the summer of 1962 ran a sparker and magnetometer survey over the lease area. The sparker records showed a much steeper base rock profile than had been extrapolated, with a maximum depth of gravel near 300 ft. Running the sparker and magnetometer surveys simultaneously caused some electrical interference in the magnetometer records, but anomalies were recorded which indicated probable areas of magnetic black sands. The sands are always found with the placer gold, and their presence will be taken as a guide to further prospecting. The next step in the programme is sampling for evaluation and this will involve some method of drilling which will be adaptable to the environment and the requirements. The excessive depth of overburden may present a special problem, and further work in the area has been delayed on this account until the summer of 1964.

The total cost of the operation was about $44 000, which included the rental of the survey craft, a 104-ft ex-Air Sea Rescue launch, complete with instruments and Shoran survey crew. The launch was already in use at Cook Inlet so transportation charges were much less than they would normally have been in that location.

Tin-bearing Sands

Deposits of cassiterite sands in drowned river alluvium are worked by Aokam Tin, Ltd., and Tongkah Harbour Dredging Co. in an area about 5 miles off the southwest coast of Thailand, south of Bhuket Island. The deposits lie in water up to 250 ft deep. The shallower ones are being mined with both bucket ladder dredges and clamshells. Aokam Tin, Ltd., is at present mining and exploring in the area west of the Malay peninsula and plans to increase its production by building a bucket ladder dredge for working at the maximum depths (Fig. 9). Similar deposits of tin are found in Indonesia and in Cornwall.

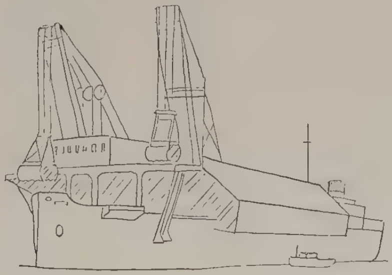


Fig. 9.—Aokam Tin's grab dredge conversion.[10]

Indonesian operations

The State Mines of Indonesia have nine bucket dredges operating offshore; eight of 14-ft^3 capacity and one of 9-ft^3, operating in 100 and 60 ft of water respectively. These are dredging in the open sea near the islands of Billiton, Singkep, and Banka, off the north coast of Sumatra.

The dredges are standard bucket ladder types working from anchored bow and stern lines, with a headline well forward of the dredge. Tailings are disposed of at the rear of the dredge and overburden and ore are worked alternately. There is only about 3 ft of freeboard and although the dredges can ride out stormy seas on the headline, work is curtailed during the monsoon season.

During 1955–58 surveys were carried out around Billiton using low-frequency Sonar. The sea bed consists of eroded granitic basement rock with marked hill and valley features. The ore has been concentrated by natural processes in the valley sediments and is rarely found outside them, so that locating the valleys eliminates large areas from immediate consideration. The sediments reach a maximum depth of 65 ft in the valleys, but on the ridges it varies from 5 to 15 ft. The Sonar survey was later checked by means of a number of bore-holes and correlation appeared to be good although sounding positions were obtained by sextant sights. The plotted bottom contours and survey results are held by the geological office at Kelapa Kampit. The maximum depth of reflection was through 150 ft of overburden in 60 ft of water; some acoustical interference was caused by using a flat-bottomed boat.

Diamond-bearing Gravels

Diamondiferous gravels occur off South West Africa along a thousand-mile coast in the area of the Orange River. It is not yet known whether the deposits are the result of continental erosion processes or if they are of offshore origin. The diamonds are found in a mixed alluvium of sand, gravel and boulders in water depths of 40–100 ft. A sample taken during 1961 from $4\frac{1}{2}$ tons of material was said to have yielded 45 diamonds with a total weight of 9 carats and a value of $450 dollars. The Collins group of companies have a 384-mile concession stretching from the mouth of the Olifants River on the Cape Coast to Laderitz in South West Africa. The concession area is defined by the southern African governments as extending from the low-water mark to 3 miles out to sea. The companies involved are: the Southern Diamond Corporation (of which Anglo American, General Mining and Finance Corporation and Middle Wits are minority shareholders); the Marine Diamond Corporation and the Establishment Collins International Vaduz, a Texas company. Four smaller concessions covering an 8-mile stretch north of the Orange River are held by the Atlantic Diamond Corporation, and further concessions are soon to be exploited by Diamond Mining and Utility Company.

There has also been Russian activity in this area during the past three years. The South African government is now preparing legislation which will extend the present 3-mile territorial limit to 12 miles.

The Collins operation

S. V. Collins began prospecting the area in 1961. Teams of coastal surveyors, equipped with the Tellurometer and Hydrodist electronic distance and position fixers, progressed with the marine prospecting units up the coast, plotting to a margin of less than a fathom. Air photographic survey helped to produce accurate mining maps of the ocean bottom and the exact areas of the diamondiferous gravel trapped in old marine terraces and submerged river beds which are invisible on land, but still exist offshore. The first sea-going mining unit was launched by Southern Diamond Corporation in May 1962 and the operation was so successful that additional dredges are already planned. The diamonds are recovered from the sea floor by means of airlift pumps, using water jets to loosen the host material. The dredged material is forced up large-diameter rubber pipes and the oversize is screened off and returned to the sea. Beneficiation, carried out on the mining barge, is by heavy medium separation followed by screen washing and conditioning with caustic soda and fish oil. The diamonds are separated from the concentrate by passing the material over grease belts prior to hand-sorting.[11]

Some difficulties have been reported to date in cleaning to bedrock, but even so, daily recoveries of 700 carats of gem-quality stones valued at $35 000 are being achieved with an operating and overhead cost of around $7000.

Iron Sands

Iron sands are known to exist offshore in many parts of the world. They have their origin in the erosional breakdown and transport of iron-bearing rocks either on land or offshore. Most of the known deposits occur in maximum water depths of about 100 ft. In Japan, offshore mining of iron sands has been carried out for many years in the southwest tip of Kyushu Island, and the fine ore was originally used for making Japanese swords. Now the production of several small operations, including the Toho Metal Mining Co., is sent to the smelters of the Yawata Iron and Steel Co. Eight other deposits of titaniferous iron sands are reported and the Yawata Co. has blocked out nearly 40 000 000 tons of refinable iron reserves in Ariake Bay.

Yawata's Ariake Bay operations

The Ariake sand deposits assay approximately 56 per cent iron, 12 per cent titanium oxide and 0·3 per cent phosphorus. Mining control and assessments are made by magnetometric surveying and coring. Various methods have been tried for winning the sand-iron from the sea floor, including hydraulic dredging, bucket ladder dredging and grab bucket excavation, the last having proved the most successful in view of the rapid currents prevailing in the bay. A pilot dredge with a $10\frac{1}{2}$-yd^3 bucket has been designed and built for operation (Fig. 10), present production with this dredge being reported as 30 000 tons/month; costs are reduced by utilizing the waste sand from the separation process for reclamation work on neighbouring sea shores.

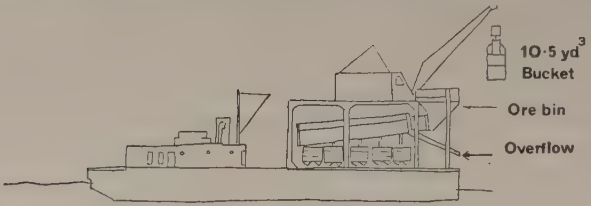


Fig. 10.—Yawata's grab dredge, Ariake Bay.[12]

Cement Sands

Raw materials for cement manufacture are mined from the sea in the form of shell sands and oyster shells. The total supply of sand for the Iceland Cement Works is mined by that company from a deposit in Faxa Bay. Oyster shells are used as the raw material for cement by the Ideal Cement Company operating in San Francisco Bay and other deposits are mined in Louisiana and Washington. These shells, which occur in beds up to 20 ft thick, are mined with hydraulic dredges and grab buckets. In San Francisco a 750-yd^3/h suction dredge is used at a maximum depth of 35 ft. Operations have been continuous for 25 years and reserves are sufficient to meet all requirements in the foreseeable future.

Faxa Bay operation

Deposits of suitable shell sand, 10 miles offshore in Faxa Bay, southwest of Akranes, were disclosed by a systematic survey of raw materials for the manufacture of Portland cement in Iceland during 1949. The mode of occurrence of these deposits is interesting. To the west of the deposits the sea bottom is shallow and rocky with a rich growth of marine molluscs. During the winter the heavy seas from the Atlantic remove, abrade and transport the dead animal shells in an easterly direction to deeper water where they again settle to the bottom. The sand makes a slope about 800 to 1200 yd wide from the point of origin to a depth of 130–140 ft. On the eastern side of the slope the tidal current runs from north to south and carries the sand particles to the south. The gradient of the slope, which is 10–12 miles long, is balanced, and the sand is constantly replenished.

The extent and composition of the deposit was initially determined by using a bottom-jaw sampler and the thickness of the deposit, which is underlain by gravel or solid rock, was obtained by using a Kullenberg core sampler—at that time a relatively new device.

The raw material as dredged is a mixture of shell particles and basaltic tuff and assays about 80 per cent calcium carbonate. The extent of the reserves that were reported from the first survey, and the fact that they are self-renewing, make periodic measuring unnecessary. The deposit is free-flowing and ranges in thickness between 3 and 13 ft at a maximum depth of 135 ft. Normal navigational methods are used for fixing the position of the dredging vessel.

Mining is done by a hydraulic suction dredge without cutter head (Fig. 11), the suction line being 24 in. in diameter and 160 ft long. The maximum limit of dredging is 140 ft and hydraulic controls permit the suction head to remain on the bottom under a constant pressure through a 6-ft rise or fall of the vessel. The capacity of the pump is 8000 tons of water per hour, carrying 3–5 per cent solids. The dredge is a sea-going vessel of 1100 tons gross weight and carries 1000 yd^3 of sand, which it transports and pumps, using the dredge pump, to stockpiles at the plant site, half a mile from the quay.

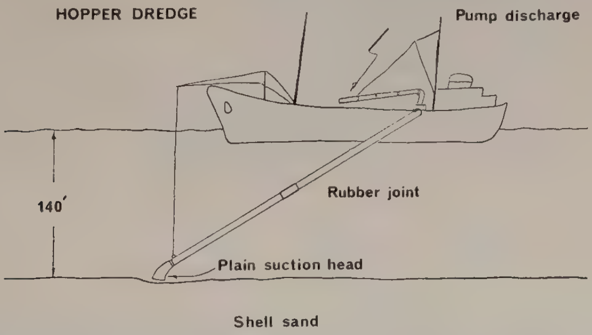


Fig. 11.—Suction dredge used by the Iceland Government cement works.[3]

Sand and Gravel

Marine and lake deposits of sand and gravel, which are generally well sorted, are commonly mined. Most operations are small and supply local markets for building materials and highway construction. Of the four types of floating equipment used in the sand and gravel industry—hydraulic dredges, bucket-ladder dredges, clamshells, and dipper dredges of the mechanical shovel type—the last are severely limited in digging and dumping range, but are useful in semi-consolidated materials. The screening and washing plant is generally an integral part of the dredge, which may process up to fifteen sizes of gravel. The material is either conveyed or pumped ashore but, where ocean-going vessels are used, it is stored in hoppers below decks.

Lorain Elyria sand and gravel

In Lake Erie the U.S. Corps of Engineers has designated the Vermillion–Lorain inner and outer areas, 6 and 10 miles square respectively, as commercial deposits of sand and gravel. These areas are worked by the Lorain Elyria Sand Co., whose plant is situated at Lorain, Ohio, about 10 miles distant. The deposits consist of clean gravels, about 50 per cent $+\frac{1}{4}$-in, and they lie at a maximum depth of 55 ft. The dredging vessel is of the sea-going hopper type, with a 12-in suction line and a capacity of 450 t/h. Screening is carried out on the vessel and the oversize is rejected overboard. At the dock the load is removed by clamshell and suction pumps to the stockpile.

Various working areas are selected by the captain on the basis of quality of material and size and gradation, and these areas are marked with buoys. Selected beds are mined according to need, and check samples are obtained by trawling during dredging.

Gravel from the English Channel[14]

It has been estimated that there are some 26 000 square miles of accessible sand and gravel deposits in the English Channel. The use of sea-going dredges to win gravel from this area is developing and the recent passing of the Continental Shelf Act by Parliament is likely to stimulate this activity further.

Specially designed vessels extract the gravel using centrifugal pumps and suction pipes. Cargoes of 500 tons are loaded in 2 hours, these being off-loaded at the dockside by means of grabs. The gravel is rounded and angular, and consists entirely of flint. The grain size of the material on the sea bottom has been shown to be related to the current in the water above it.

Sea-floor Nodules

Proposed methods of mining manganese and phosphorite nodules include deep-sea dredging, hydraulic dredging (Fig. 12), or clamshell bucket dredging. Although no companies are actively engaged in mining the nodules, feasibility studies indicate that they are potentially economic.[13] Preliminary exploration and development studies have reportedly been carried out by American companies during 1962 at a cost of over half a million dollars, and it is anticipated that this expenditure will have been increased during 1963. Kennecott Copper Corporation and Newport News Shipbuilding and Drydock Company are both

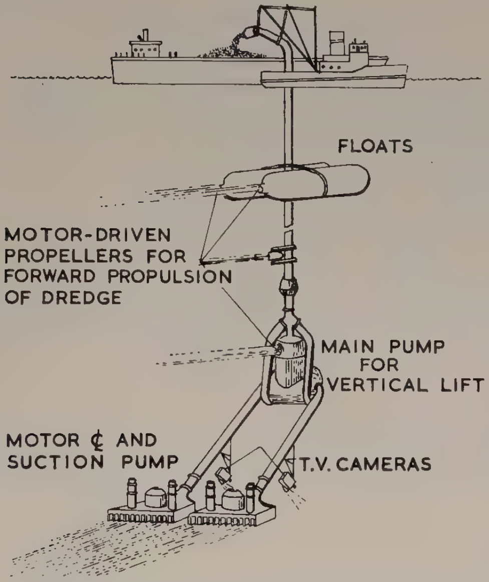

FLOATS
MOTOR-DRIVEN PROPELLERS FOR FORWARD PROPULSION OF DREDGE
MAIN PUMP FOR VERTICAL LIFT
MOTOR ₵ AND SUCTION PUMP
T.V. CAMERAS

Fig. 12.—Proposed design for a suction dredge to gather nodules from the deep-sea floor.[13]

reported to be going ahead with plans to build dredges to mine manganese nodules at depths of at least 4000 ft.

REFERENCES

1. Mero, J. L. *Manganese nodules on the ocean floor*. Report for Jackling Fellowship Committee, I.M.R., Univ. of California, Berkeley, Aug. 6, 1961.

2. Beckman, W. C. Geophysical surveying for a channel tunnel. *New Scientist*, **7**, no. 175, March 24 1960, 710–2.

3. Cruickshank, M. J. The exploration and exploitation of off-shore mineral deposits. M. S. Thesis, Colorado School of Mines, Sept. 1962.

4. Peele, R. *Mining engineers' handbook, vol. I.* (N.Y.: John Wiley & Sons, Inc., 1941).

5. Romanowitz, C. M. Private communication, 1963.

6. Saunders, D. W. Design for a grab dredge. *Min. Mag., Lond.*, **106**, May 1962, 272–9.

7. Low, D. W. Dredging craft. *Engineering, Lond.*, **173,** March 21 and 28 1952, 377–9; 409–11.

8. Shalowitz, A. L. Shore and sea boundaries. *U.S. Coast and Geodetic Survey Publ.* 10–1 (Washington, D.C.: Govt. Printing Office, 1962).

9. Alaska, State of. *The State Lands Act, 1959.* Article IX, Chapter 123. Session Laws of Alaska, 1961.

10. Romanowitz, C. M. The dredge of tomorrow. *Engng Min. J.*, **163,** April 1962, 84–91.

11. Collins, S. V., and Keeble, P. Diamonds from the sea bed. *Underwater yearbook, 1962.* (Chicago: Underwater Society of America, 1962), 12–4.

12. Grab dredge will mine sand iron. *Engng Min. J.*, **163,** April 1962, 94.

13. Mero, J. L. Economics of deep-sea mining. *Min. Congr. J.*, **47,** Sept. 1961, 52–6, 68.

14. Batt, V. G. W. Gravel fields in the sea bed. *Proceedings of the Sand and Gravel Association short course* (London: Sand and Gravel Association, 1963), 63–9.

Appendix

In the Appendix tabular data on mining dredges, current world-wide offshore mining operations (together with a location map), unconsolidated deposits offshore (with commodities and locations), and market prices (as at August 1963) of metals and ores found in offshore alluvial deposits are given.

TABLE I.—*Data on mining dredges*

Company	Location	Mineral	Depth ft	Capacity	Capital cost	Power	Production	Unit cost
BUCKET-LADDER DREDGE								
Tongkah Harbour	Thailand	Cassiterite	155	15 ft^3	—	—	250 yd^3/h	19.4 cents/yd^3
Aokam Tin, Ltd.	Bhuket Island	Cassiterite	60–70	14 ft^3	—	—	200 yd^3/h	28.3 cents/yd^3
State Mines	Indonesia	Cassiterite	60–100	14 ft^3	$2 500 000	—		
DEEP-SEA DRAG DREDGE								
Proposed	—	Nodules	1000	13 t	—	1502 hp	658 t/d	$11.50/t*
,,	—	,,	3000	,,	—	1600 hp	431 t/d	$11.70/t
,,	—	,,	5000	,,	—	1720 hp	326 t/d	$23.60/t
CLAMSHELL								
Yawata Iron & Steel	Japan	Iron sands	90	10·5 yd^3	—	—	—	$5.30/t
HYDRAULIC HOPPER DREDGE								
U.S. Corps of Eng.	United States	Sand and clay	55	2 × 20 in.	$636 000	1250 hp	2040 yd^3/h	13.0 cents/yd^3
Alum. Co. of America	Surinam	Over-burden	45	1 × 24 in.	$800 000	3200 hp	2000 yd^3/h	19.6 cents/yd^3
DEEP-SEA HYDRAULIC DREDGE								
Proposed	—	Nodules	1000	1 × 20 in.	$4 500 000	778 hp	209 t/h	$2.29/t
,,	—	,,	5000	,,	$5 100 000	3890 hp	,,	$2.85/t
,,	—	,,	10 000	,,	$5 900 000	7780 hp	,,	$3.55/t

* Nodule concentration of 1 lb/ft^2.

TABLE II.—*Current world-wide offshore mining operations (1963)*

(See Fig. 13 for locations)

Location	Mineral	Status	Organization	Mining method(s)	Water depth	Distance from shore	Capacity	$ Value of operation (estimated)
1. Norton Sound Alaska	Gold	Exploration	Shell Oil Co.	Not selected	60′ (Up to 300′ of over-burden)	0–1 mile	N/A	Exploration cost to date approximately $50 000
2. Stephens Passage Alaska	Gold	Planning	Williams Hydraulics Co.	Hydraulic grab proposed	520′–900′	0–3 miles	N/A	$1 000 000 alleged to have been invested since 1933
3. Nome Teller Tin City Platinum Kodiak Juneau Alaska	Primarily gold (also silver and platinum)	Promotion	39 Permits granted	Not selected	60′–150′	0–5 miles	N/A	$1.00/acre (State of Alaska lease-hold charges)
4. Bhuket Island Thailand	Tin-bearing sands	Active	Aokam Tin, Ltd.	Bucket-ladder dredges & clamshells	Max: 120–130′ Normal: 60–70′	5 miles	200 yd^3/h	$500 000/yr
5. Bhuket Island Thailand	Tin-bearing sands	Active	Tongkah Harbour Dredging Co.	Bucket-ladder dredges & clamshells	Max: 155′ Normal: 155′	1 mile	250 yd^3/h	$500 000/yr
6. W of Malay Peninsula	Tin-bearing sands	Active plus additional exploration	Aokam Tin, Ltd.	Bucket-ladder dredges & clamshells	Unknown	Unknown	Unknown	Unknown
7. Indonesia (off NE Coast Sumatra)	Tin-bearing sands	Active	State Mines of Indonesia	Bucket-ladder dredges	Max: 70′	1–6 miles	2000 yd^3/h (aggregate)	$1 000 000 yr

TABLE II (*continued*)

Location	Mineral	Status	Organization	Mining method(s)	Water depth	Distance from shore	Capacity	$ Value of operation (estimated)
8. South West Africa	Diamondiferous gravels	Active	Collins Group of Companies: Southern Diamond Corp., Marine Diamond Corp., Collins Int'l. Vaduz	Water jetting airlift pumps	30′–100′	0–3 miles	700 carats/day	$30 000/day
9. Mozambique Channel, Southeast Africa	Diamonds	Equipment construction	Ocean Science & Engineering Co. under contract from De Beers Mines, Ltd.	Airlift & suction combination	Less than 200′	0–5 miles	Unknown	$1 000 000 committed for equipment (alleged)
10. South West Africa	Diamondiferous gravels	Exploratory mining	Russia	Unknown	To 300′	Beyond 3 mile limit	N/A	N/A
11. Ariake Bay Japan	Iron sands	Active	Yawata Iron & Steel	Grab buckets & suction	100′ max.	1 mile	30 000 t/month	$300 000/month
12. Southwest Tip of Kyushu	Iron sands	Active	Toho Metal Mining Co. & other small firms	Suction dredging	30′–100′	0–1 mile	3000 t/month	$30 000/month
13. Faxa Bay SW of Akranes	Cement sands	Active	Government of Iceland	Suction dredge w/o cutter head	Max: 135′	10–12 miles	200 t/h	$200 000/yr

TABLE II (*continued*)

Location	Mineral	Status	Organization	Mining method(s)	Water depth	Distance from shore	Capacity	$ Value of operation (estimated)
14. San Francisco Bay	Oyster shells	Active	Ideal Cement Co.	Hydraulic dredges & grab buckets	35′	0–½ mile	750 yd^3/h	$2 000 000/yr
15. Louisiana, Alabama	Oyster shells	Active	Lake Charles Tower & Dredger Co. Southern Oyster Shell Milling Corp.	Suction dredges & clamshells	Up to 50′	0–1 mile	400 yd^3/h	Less than $3 000 000/yr
16. Multiple locations in U.S. (Harbour Areas & Mid Pacific U.S. possessions)	Sand and clay	Active	U.S. Corps of Engineers	Hydraulic hopper dredge	55′	0–5 miles	110 000 000 yd^3/yr aggregate (20 dredges)	$12 000 000/yr (Aggregate cost)
17. U.S.A.—40 miles west of San Diego	Phosphorite	Suspended	Collier Carbon & Chem. Co.	Unknown (suspected to be wire-line drag-dredge)	600′	40 miles	500 000 t/yr	$6 000 000/yr (Potential)
18. Mid-Pacific near Tahiti	Manganese nodules (Hi-nickel/cobalt)	Construction of equip.	International Nickel & Newport News Shipbuilding (form of association unknown)	Suction dredge w/ submerged pumps	4000′	Out of sight of land	1 250 000 t/yr	$75 000 000/yr (Potential)
19. Unknown	Manganese nodules	Equipment designed (bids asked)	Kennecott Copper Corp.	Suction dredge w/ submerged pumps (presumed)	Deep ocean to 12 000′	Out of sight of land	Unknown	Potentially large

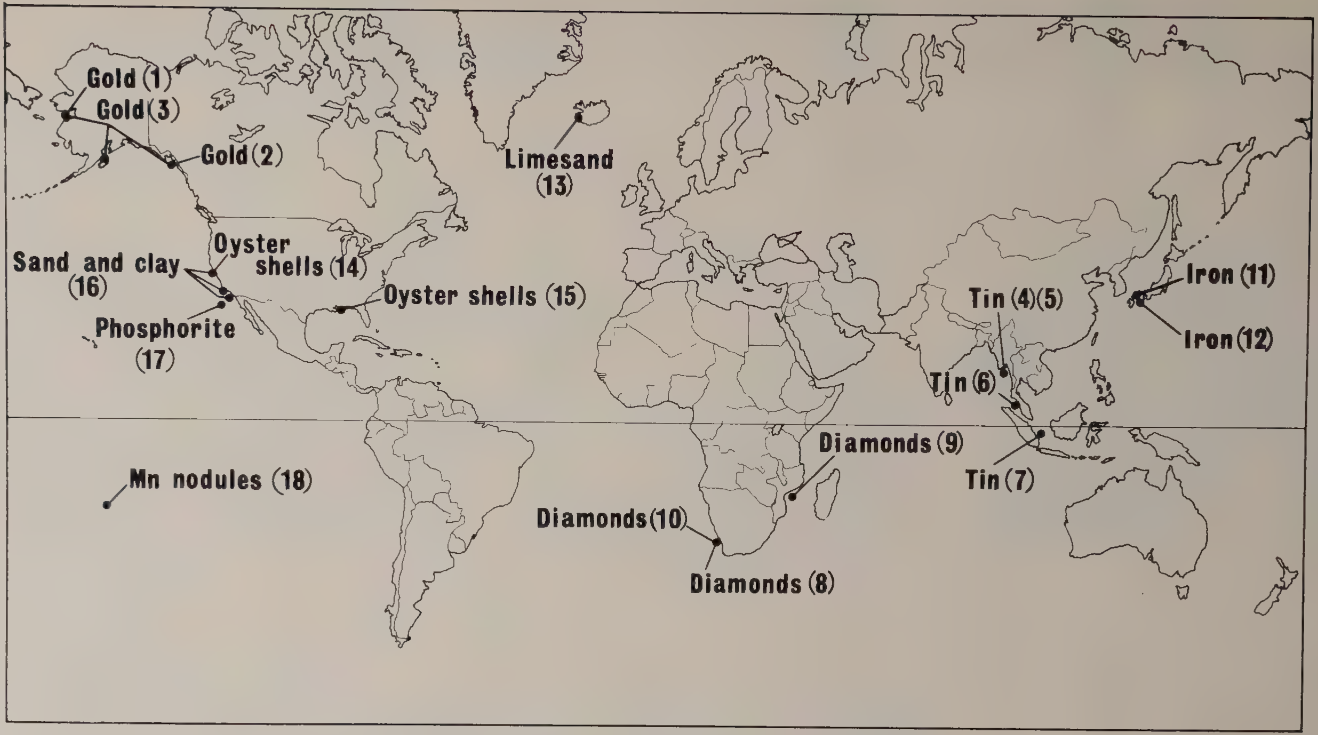


Fig. 13.—Location of operations referred to in Table II.

TABLE III.—*Unconsolidated deposits offshore, known and/or operating, showing commodities*

United States of America	
Au	Alaska (Nome, Kodiak Island, Yakalaga), Washington, Oregon/California border
Ag	Washington, Oregon/California border
Pt	Washington, Oregon/California border
Mineral sands	Washington, Georgia, Florida, Alabama, North Carolina
Fe	Columbia River, Oregon/California border, California (Redondo Beach)
Ti	Columbia River, Oregon/California border, Delaware
Garnet	Oregon/California border
Phosphorite	California
Other parts of the world	
Diamonds	South West Africa
Au	Chile
Mineral sands	Costa Rica, Denmark, Ceylon, Philippines, Northern Territory (Australia), Tasmania, New Zealand, Papua and New Guinea
Sn sands	Indonesia, Thailand, Cornwall (England)
Fe	Chile, Japan
Ti	British West Indies, Mozambique, South Africa
Mn	Costa Rica
Ca sands	Iceland, Fiji
Pyroxene sands	Fiji
Phosphorite	South Africa

TABLE IV.—*Unconsolidated deposits offshore, known and/or operating, showing locations*

United States of America	
Alabama	Mineral sands
Alaska	Au (Nome, Kodiak Island, Yakalaga)
California	Au, Ag, Pt, Fe, Ti, Garnet (Oregon border); Fe (Redondo Beach); Phosphorite (Southern California)
Delaware	Ti
Florida	Mineral sands
Georgia	Mineral sands
N. Carolina	Mineral sands
Oregon	Au, Ag, Pt, Fe, Ti, Garnet (California border); Fe, Ti (Columbia River)
Washington	Au, Ag, Pt, Mineral sands
Other parts of the world	
Africa	Diamonds, Ti, Phosphorite (South West Africa)
Britain	Sn
B. W. Indies	Ti
Ceylon	Mineral sands
Chile	Au, Fe

TABLE IV (*continued*)

Costa Rica	Mineral sands, Mn
Denmark	Mineral sands
Fiji	Ca sands, pyroxene sands
Iceland	Ca sands
Indonesia	Sn
Japan	Fe
Mozambique	Ti
N. Zealand	Mineral sands
Northern Territory, Australia	Mineral sands
Papua and N. Guinea	Mineral sands
Philippines	Mineral sands
Tasmania	Mineral sands
Thailand	Sn

TABLE V.—*Market prices of metals and ores found in offshore alluvial deposits* (*Engng Min. J.*, August 1963; 7.15 s. = $1)

Precious metals	
Gold	250 s./oz
Platinum	555 s./oz
Metallic ores	
Beryl	242 s./s.t.u. BeO, 10–12% BeO
Cassiterite	90s./lb Sn
Columbite–Tantalite	6.4–7.2 s./lb, 10:1, 65% Cb_2O_5, Ta_2O_5
Chromite	178–258 s./l.t., 48% Cr_2O_3
Ilmenite	171 s./l.t., 59% TiO_2, f.o.b.
Magnetite	72 s./l.t., 51% Fe
Monazite	0.7–1.1 s/lb, cif, Re oxide 55%
Rutile	728 s./s.t., 94% TiO_2
Scheelite	55–61 s./s.t.u., WO_3, 65%
Wolframite	55–61 s./s.t.u., WO_3, 65%
Zircon	357 s./l.t., 65% ZrO_2
Industrial minerals	
Barite	86–114 s./s.t.
Garnet	178–1000 s./s.t.
Phosphorite	50–72 s./l.t., 66% f.o.b. Florida
Sand and Gravel	21–36 s./s.t.
Muscovite	171 s./s.t. scrap
Diamonds	355 s./carat

622.234.8:622.342.13 + 622.349.1(86 + 84) *Paper 8*

Gold and Platinum Dredging in Colombia and Bolivia, South America

PATRICK H. O'NEILL, B.Sc.(Min. Eng.), E.M., M.I.M.M.
Chairman, Pato Consolidated Gold Dredging, Ltd., and Executive
Vice-President, International Mining Corporation, New York, U.S.A.

SYNOPSIS

Alluvial mining has been carried on by various methods in practically every country in South America, but the greatest output has come from Colombia, gold being the principal product, with platinum in appreciable quantities.

The paper briefly describes current dredging operations in the three main areas in Colombia, with some reference to exploration by drilling, costs and difficulties, and development drilling, including Peru and Bolivia. There is a comment on the evaluation of prospecting results.

A section on titles and concessions discusses royalty terms in Colombia and Bolivia.

The high cost of equipping a property in South America is illustrated by an example from Bolivia.

Tables give general information and operating data from four operating companies, and dredge operating costs. The operating conditions are described and recovery systems, sales and future outlook are mentioned.

Although alluvial mining has been carried on by various means in practically every country throughout South America, the major production has come from Colombia which has coastal areas on both the Caribbean and the Pacific.

There are three main areas of present-day alluvial mining in Colombia. From north to south they are the Nechi River area, in the vicinity of Zaragoza, approximately 7° 30′ north of the equator, in the Department of Antioquia. The Nechi flows into the Cauca River and it, in turn, flows into the Magdalena River, the principal artery for river transport to the interior of Colombia. Pato Consolidated Gold Dredging, Ltd., a Canadian company, operates in the Nechi River area.

The second area, southward, is in the Department of Choco in the vicinity of Istmina, Condoto and Novita, approximately 5° N latitude. Present operating areas are on a tributary of the San Juan River, a southward-flowing river that enters the Pacific just north of Buenaventura, the principal Colombian port on the Pacific Coast. One dredge operated on a tributary of the Atrato River in this area from 1932 until 1944, when it was decided to move it to the San Juan drainage

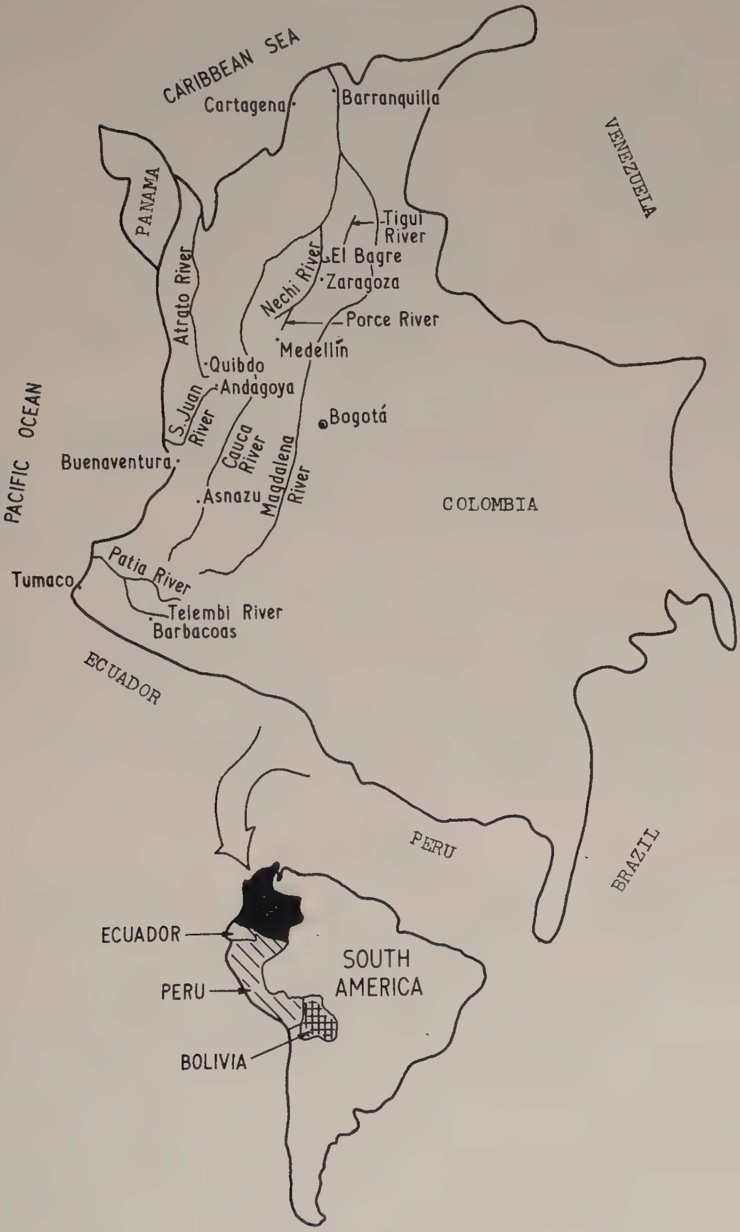


Fig. 1.—General map showing areas referred to.

area to the south. The Atrato flows northward to the Caribbean and although the dredge had only to move 40 air miles from one drainage area to the next, it was cheaper to move it 1200 miles by river and ocean through the Panama Canal after it completed dredging in the Atrato watershed. Farther south and very close to the Ecuador border at approximately 1° 45′ N latitude is the placer district near Barbacoas on the Telembi River and its tributaries in the Department of Narino.

There has been some dredging in other locations in Colombia, two of which were on widely separated tributaries of the Cauca River—Supia, just south of Medellín, and Asnazu, south of Cali. There was also one dredge working for a few years on the headwaters of the Patia River, south and west of Popayán. Asnazu, the most important of these areas operated one to two dredges from 1935 to 1956 and produced $15 531 000 in gold from 82 828 000 yd^3 of gravel.

Gold is the principal product of the alluvial deposits of Colombia, but platinum is produced in appreciable quantities in the Choco and in very minor quantities in the other areas. Russia was the largest producer of platinum until just before World War I. Colombia was the principal world producer from 1916 to 1923 and although her production of platinum has remained fairly constant since 1917, it now constitutes less than 5 per cent of world production.

Production of gold and platinum in Colombia continues to be appreciable, mainly in the three principal mining areas. This paper will be limited to the operations of the present-day dredging companies in Colombia with some reference to drilling and exploration costs in Peru and Bolivia and to costs of equipping and operating a single gold dredge operation in Bolivia.

EXPLORATION

The earlier prospecting in the Choco area was done with Empire or Union drills and in the Pato area by Empire drills and the steam-driven Keystone drills. The main drawback to the Empire drill, especially in the Choco area where all the early dredging was done in rivers, was that it could not be satisfactorily used in rivers that flooded frequently. The platform of the Empire drill screws on to the drive pipe and a hole would often be started near the river. The river would then rise as much as 10 ft or more in a few hours, forcing work to be abandoned, and often the tools and equipment would be lost.

In the early 1920s William F. Ward, an engineer who had considerable prospecting and dredging experience, developed for Compañia Minera Choco Pacífico a hand drill, known as the Ward drill, that could be used in a river on a barge or on a platform built on two large canoes. It was also easily portable and excellent for drilling on land and proved to be much more efficient for drilling coarse gravel than the Empire drill. Greater depths could also be achieved.

The Ward drill (Fig. 2) is a simple unit that can be set up quickly. Basically, it consists of four straight poles, 5–7 in. in diameter at the large end, which are set into notches in planks to prevent their sinking into the ground. The poles are joined at the top by a shaft that holds the pulley for the drill wire or rope. The walking beam is activated by 8–10 men lining up on the cross arm. They pull down to raise the tools and vary their manner of movement depending on whether they are driving casing, drilling, or pulling casing. The Ward drill is logically most efficient in shallow ground, but can be—and frequently is—used in deeper ground, depths of 90 ft not having been uncommon in the Pato operations.

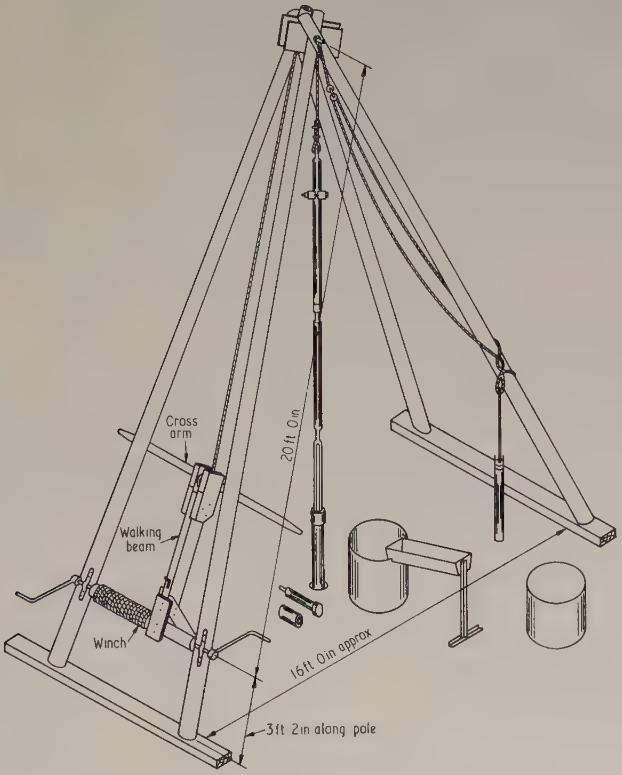


Fig. 2.—Diagram of Ward drill.

The Bucyrus and Keystone track-mounted power churn drills are also used wherever ground conditions and drilling objectives permit. Besides being generally more economical on a cost-per-foot basis, the principal advantage of the power drills is the larger sample obtained. With power drills 6-in diameter casing is used, whereas 4- or 4½-in diameter is used with hand drills.

Cost per foot of drilling will vary widely depending on conditions encountered. The type of material to be drilled is usually the most important condition affecting the cost but location, terrain, experience of the crew, supervision, condition of the equipment, supply line and many other factors also determine the cost per foot. Tables I–IV show recent years' statistics on drilling done in the four dredging operations covered in this paper. As can be seen, cost per foot for hand drilling varies from 78 cents under exceptionally favourable circumstances near the main camp in Choco to an abnormally high figure of $20.14 per foot in an extremely isolated area that had to be supplied by plane from the Pato camp. Without previous experience in any given area it is impossible to estimate within any reasonable limits the footage or cost per foot that might be obtained per day.

A total cost per month per drill is a better figure to use, especially for exploratory work in a new area, in estimating costs of an exploration programme. When costs of an exploration programme in Peru in 1959 were estimated, figures of $5000 per month for a single-drill operation and $7000–$8000 per month for a two-drill operation were used. These cost figures are high, but the area was very remote. Initial supply lines required packing in by mules or llamas down the eastern slope of the Andes while an airstrip was built by hand.

The exploration programme along the Inambari and the Madre de Dios Rivers and several of their tributaries lasted a full three years. Six airstrips were constructed, all by hand work with homemade wheelbarrows. Equipment for five drills was purchased new and this was supplemented by additional 4½-in casing and tools to replace those lost or damaged. Some of the areas were fairly shallow—20 to 45 ft—but the major part of the time was spent on the Inambari where depths ranged from 80 to 163 ft. From one to three Ward drills were in operation all the time.

The total cost for this entire programme averaged $7000 per month. The wages paid to labourers on this programme varied from 20 to 25 soles per day in basic pay and the total cost, including social benefits, was 35–40 soles or approximately $1.50 per day. In addition, it cost 20–25 cents per day to supply each man his usual requirements of sugar, rice, dried mutton or llama, etc. On each drill 14–16 men were employed, with additional labour when necessary for airfield construction, clearing the jungle for drill sites and packing in supplies.

Wages in Colombia for drill workers have been somewhat higher

TABLE I.—*Comparative drilling data—Compañía Minera Choco Pacífico, S. A.*

Year	Location	Type of drill	No. of holes	Depth, ft			Average ft/shift	Cost/ft U.S. $	Remarks
				Max.	Min.	Av.			
1959	Chiqui Choqui, land	BE 6″ power	24	60	36·0	50·8	18·0	1·11	Deep ground, some swampy
1959	Suruco Area ,,	,,	17	59	16·0	37·9	21·0	1·44	Swampy ground, drill moving slow
1960	Upper Tamana, water	Ward 4″ hand pontoon	25	25	4·0	17·7	7·0	5·38	High rivers, hard drilling, canoe trans.
1960	Rio Iro Area, land	Ward 4″ hand	305	31	4·0	14·2	24·0	2·63	Canoe trans., built camp, high rivers
1960	Novita Val. ,,	,,	121	30	6·0	11·6	39·0	1·18	Good trans., soft ground, used est. camp, swampy
1961	Novita Val. ,,	,,	160	48	12·0	28·8	35·0	1·80	Built camp, swampy ground, easy drilling
1961	Opogodo Val. ,,	,,	357	36	7·0	16·7	41·0	1·19	Shallow ground, easy moves, good trans.
1961	Garrapatas ,,	,,	18	39	14·0	25·6	14·0	8·77	Built camp, high rivers, difficult trans.
1961	Raspadura ,,	,,	35	71	19·0	47·7	24·0	1·88	Deep ground, very slow after 50 ft
1962	Raspadura ,,	,,	76	105	10·0	60·8	21·0	1·84	—do.—
1962	La Ventura ,,	,,	28	49	17·0	32·0	22·0	1·69	Canoe trans., much time lost travelling
1962	Bajo Opogodo ,,	,,	50	42	12·8	22·2	38·0	2·10	Good trans., no drilling problems
1962	El Banco ,,	,,	262	65	15·0	32·1	37·0	0·78	—do.—
1963	El Banco ,,	,,	261	83	10·5	39·1	34·0	1·98	—do.—

TABLE II.—*Prospect drilling data—Pato Consolidated Gold Dredging, Ltd.* Ward: 13 Men; Keystone: 5 to 8 men

Year	Location	Type of drill	No. of holes	Depth, ft.		Average ft/shift	Cost/ft U.S. $	Remarks
				Max.	Av.			
1960	Carabobo	Hand drill	8	90·0	78·3	20·6	3·40	Scout drilling
	Carabobo	,, ,,	20	95·0	84·1	14·0	3·67	Development drilling
	Tenerife	,, ,,	11	82·0	65·3	16·4	3·03	,, ,,
	Boyaca	Keystone power drill	33	95·0	73·8	22·1	2·65	,, ,,
	Colombia	,, ,, ,,	4	77·0	72·0	26·0	1·96	,, ,,
	Tenerife	,, ,, ,,	24	96·0	65·3	22·6	3·03	,, ,,
1961	Boyaca	Hand drill	4	93·0	92·0	14·2	2·52	Development drilling
	Tenerife	,, ,,	16	94·0	70·7	21·2	2·54	,, ,,
	Carabobo	,, ,,	7	100·0	80·6	12·4	2·66	,, ,,
	Carabobo	Keystone power drill	12	83·0	80·6	20·6	2·66	,, ,,
	La Quiebra	Hand drill	8	90·0	80·1	14·7	2·83	,, ,,
	Tenerife	,, ,,	6	81·0	58·7	18·3	4·93	Scout drilling
	Tenerife	Keystone power drill	19	90·0	70·7	20·0	2·54	Development drilling
1962	Madreseca	Hand drill	41	36·0	25·2	31·3	2·95	Scout drilling, extremely isolated areas
	San Pedro	,, ,,	4	66·0	47·5	22·2	9·99	—do.—
	Regencia	,, ,,	8	45·0	25·2	8·4	20·14	—do.—
	El Hatillo	,, ,,	4	50·0	38·8	20·3	8·78	Scout drilling
	La Quiebra	,, ,,	11	93·0	76·6	15·6	2·63	Development drilling
	Cienaga Grande	,, ,,	9	91·0	77·0	15·7	3·97	,, ,,
	Colombia mine	,, ,,	4	70·0	61·8	16·6	2·63	,, ,,
1963	La Quiebra	Hand drill	19	79·0	59·7	22·6	2·51	Development drilling
	La Quiebra	,, ,,	22	47·0	25·9	17·2	4·30	Scout drilling
	Cienaga Grande	,, ,,	10	51·0	40·3	21·9	2·91	,, ,,

TABLE III.—*Comparative drilling data—Compañía Minera de Narino*

Year	Location	Type of drill	No. of holes	Depth, ft			Average ft/shift	Cost/ft U.S. $	Remarks
				Max.	Min.	Av.			
1960	Guelmambi, land	Ward 4″ hand	33	42·0	4·3	23·8	12·7	2·93	Hard drilling, boulders
1961	Guelmambi, land, water	”	111	88·0	4·5	32·6	10·8	3·35	—do.—
	Inguambi, land	”	34	52·0	9·0	24·9	17·6	1·94	Soft drilling
	Tanche, land	”	162	41·0	6·0	22·3	13·7	7·94	New camp, swampy ground
	Cucarachera, land	”	30	66·0	8·0	28·7	17·9	1·73	Normal drilling, trans. good
1962	Tanche, land	”	112	48·0	11·0	27·7	16·2	2·98	Swampy ground
	Guelmambi, land, water	”	116	68·0	5·6	28·1	11·0	4·70	Hard drilling, boulders
	S. Ana, land	BE 6″ power	9	106·0	77·0	92·1	31·4	6·32	Swampy ground, drill frequently under water
	Nambi, land	Ward 4″ hand	10	48·5	19·0	32·7	32·7	4·13	Canoe transportation
	Upp. Telembi, land, water	”	40	43·0	5·0	19·2	8·6	4·78	Drill balsa, high rivers
1963	Guelmambi, land, water	”	172	56·0	4·3	25·4	7·6	7·56	Hard drilling, boulders
	S. Ana, land	BE 6″ power	48	130·0	29·0	86·1	14·4	5·20	Swampy ground, drill frequently under water
	La Humildad, land	”	28	146·0	19·0	99·6	23·2	3·41	—do.—
	S. Francisco, land	”	4	105·0	89·0	98·0	21·8	2·44	—do.—
1964	Guelmambi, land, water	Ward 4″ hand	25	44·0	5·6	23·2	7·3	9·26	Social benefits, liquidation of personnel, canoe transport
	La Humildad, land	BE 6″ power	12	134·0	67·0	111·1	18·5	1·65	Normal drilling
	S. Francisco, land	”	13	101·0	22·0	71·6	19·4	2·13	—do.—
	S. Jamie, land	”	9	139·0	60·0	97·7	24·4	1·83	—do.—
	Lower Telembi, land	”	9	98·5	62·0	73·7	13·8	7·72	Playas, frequently flooded

TABLE IV.—*Comparative drilling data—Bolivia*

Month	Location	Type of drill	No. of holes	Depth, ft			Average ft/shift	Cost/ft U.S. $	Remarks
				Max.	Min.	Av.			
1961									
Jan.–Mar.	Challana, land	Ward 4½″	8	65·0	24·7	41·4	7·8	9·40	Boulders—hard drilling
June	Teoponte, land	BE 6″	4	84·0	32·0	57·3	8·4	4·30	,,
July	Sitahuara, land	Ward	3	51·0	26·0	39·7	8·0	9·80	,,
Aug.	Sitahuara, water	,,	6	51·0	11·0	27·9	5·7	8·70	,,
Sept.	Teoponte, land	BE	4	111·5	13·5	70·8	12·9	2·90	,,
Oct.–Dec.	Teoponte, land	,,	13	121·0	49·0	68·8	16·2	2·60	,,
1962									
Jan.–Feb.	Teoponte, land	BE	14	93·0	42·0	57·8	19·1	2·90	,,
Mar.–May	Challana, land	Ward	14	57·0	21·0	30·2	6·1	6·70	,,
June	Duran Playa, land	,,	12	58·0	15·0	26·3	12·8	2·80	,,
1963									
Jan.–Feb.	Teoponte, land	BE	7	99·0	34·0	65·5	14·4	2·40	,,
June–Sept.	Kaka River, water	Ward	34	42·6	1·7	21·2	9·1	6·30	,,
Nov.–Dec.	Asilahuara, land	BE	12	64·0	21·0	40·1	11·8	5·90	,,
1964									
Jan.–Mar.	Asilahuara, land	BE	12	89·0	24·5	51·1	14·9	4·80	Transport to other side

in the last year or so and now vary from $1.50 to $2.00 per day in basic wages plus an additional 35 per cent in social benefits.

Development drilling with Ward drills near established camps in Colombia costs an average of approximately $2000 per month. Power drills on development work cost less because fewer men are required. Two Bucyrus Erie power drills working in Narino last year averaged approximately $2600 per month for both drills. The cost for one would probably be $1700–$1800 per month. Recent costs on an exploration programme in Bolivia near transport and supply lines were running at $2000 per month for a power drill on one shift and $3000 per month with the same drill on a two-shift basis. On a two-shift basis costs per foot were averaging approximately $5.00. Costs of supervision and logistical support are principal items which, when applied to a multi-drill programme, can result in reduced unit overheads.

Evaluation of Prospecting Results

A discussion of the technical details of prospect drilling and the calculations of drilling results are beyond the scope of this paper, but the testing and evaluating of a placer deposit is not by any means an exact science and a few comments on the evaluation of prospect drilling may be of interest.

In valuing a drill-hole there are two governing factors—the gold recovered and the volume of the material from which it was derived. The gold can be weighed and therefore is definite, but the determination of the true or original volume of material from which the gold was extracted involves many variables. There is, of course, a theoretical volume of core rise that should be obtained for each foot of drive of the casing, which is a function of the outside diameter of the casing drive shoe. This theoretical volume will vary as the drive shoe wears down and cannot be obtained in actual practice as it depends on the size and looseness of the gravel, the size of the casing, the condition of the drive shoe and the length of the drive. The volume that is extracted is a deslimed volume, and even where the slimes are settled and measured the exact volume recovered cannot be determined. There is a tendency to use positive core correction factors—that is, to increase the amount of gold by the same percentage that the core rise or volume recovered is below the theoretical. This is a dangerous practice and should not be condoned in a new area, especially when there are boulders present or the gravel is very coarse. The only case where positive corrections should be used is when considerable dredging recoveries can be compared with the prospecting results. Experience has shown that even the application of proved factors to adjoining areas can be dangerous. In Choco, for example, where positive corrections up to 42 per cent were allowed based on dredging experience, these same factors applied to deposits 10 to 15 miles away did not stand up and the resulting recoveries were very disappointing, averaging less than 60 per cent of expectancy. Negative corrections when the core rise or volume recovered exceeds the theoretical should, on the other hand, always be applied.

TITLES AND CONCESSIONS

The major part of the areas that have been dredged and that are yet to be dredged in Colombia are held under mining claims issued late in the last century or in the early 1900s. One large area, the Condoto River in Choco, was dredged under a special concession that was granted to General Castillo in 1907 for his efforts during the war of One Thousand Days. There was considerable dispute over the General's title, however, because others claimed title through mines denounced prior to the time the concession was granted to General Castillo. The General's title was upheld by the Supreme Court after several years of litigation. The mining claims were generally purchased from the owners, although occasionally they were worked on a royalty basis. A law was passed in 1937 which provided that all navigable rivers and land within one kilometre thereof on either side could only be obtained for mining under concession from the government. Under the concession a sliding-scale royalty based on the tenor of the ground dredged is payable to the government. This scale is as follows:

Gold per m^3 dredged *mg*	*Percentage of the gross proceeds exploited*
Up to 150	2
151–200	3
201–300	4
301–400	5
401–500	7
501–600	9
601–700	12
701–800	16
Over 800	20

Both when mining government concessions and privately titled mining claims certain production taxes, import duties and income taxes are payable to the government. Income taxes are at a maximum rate of 38 per cent and all taxes together total approximately 50 per cent of the pre-tax net profit.

Over the years the major problems in Colombia have been legal in nature involving title ownerships and dredging rights in navigable rivers, even where titles conferred before the passage of the new law were upheld by the courts. There have been very few problems in relation to mining under government concession. Inspectors are appointed by the Ministry of Mines to supervise the dredging, recovery, yardage measurements and reports.

The only dispute of consequence involved at one time the inclusion of transient river sand in yardage calculations for determining royalties. The rivers in Colombia rise quite often due to heavy tropical rains. These rises vary from a few feet in the large rivers in the lower Pato area to as much as 30 ft or more in the upper Narino area. Even at low stages the rivers carry considerable volumes of sand, especially at Pato, where the top part of the 90-ft dredge section is mostly sand. When the rivers rise, the volumes of sand carried into the dredge ponds require many hours to clean out. It is impossible to measure these volumes of sand accurately, but after extensive negotiations and discussions with the government mines officials acceptance of the volumes on an hourly basis as reported by the winchmen on duty was agreed. The average yardage of the dredge is then used for the calculation.

Small, narrow and swifter rivers have to be dredged on a headline digging upstream and thus suffer the full effect of transient sand or gravel in high river stages. In order to alleviate the sand digging problem in the deep ground in Pato's wide rivers the dredges dig downstream using a spud, and the only cut that is bothered with sand is the river cut where the sand washes in the side of the dredge cut.

In 1956 a contract was worked out in Bolivia on a gold placer property east of the Andes. The ground was in the government preserve and was to be worked under a concession. The government was interested in a partnership arrangement and rather than a 50 per cent split of the net profit, which would have involved accounting problems, a formula was worked out as a sliding scale based on estimated costs and production to result in the equivalent of a 50 per cent split of earnings. The contract provides for a flat 5 per cent gross royalty plus a sliding-scale sole tax on the following basis:

Gold per yd^3 dredged mg	*First 5 years per cent*	*Second 5 years per cent*	*Last 15 years per cent*
300–400	4	4	6
401–500	6	8	11
501–600	10	12	15
601–700	13	15	19
701–800	16	18	22
801–1000	19	21	25
Over 1000	22	25	30

The sole tax is in lieu of all import, export and income taxes. The company is free to export the gold, and royalty and sole tax are settled quarterly in dollars or in kind at the election of the government. Shortly after the start of the operation they selected payment in kind, which is still in force.

The contract was based on many years of experience in Colombia and the lawyer for the companies in Colombia was the principal adviser in writing the Bolivian contract.

Only one major problem has arisen in regard to the Bolivian contract and that concerns the volume or yardage used in the calculations to arrive at the tenor. In view of the previous problem with transient sand in Colombia, care was taken to cover this point clearly in the Bolivian contract. There has never been any problem in Colombia with tailings that have had to be re-dredged and the Bolivian contract clearly states that *all* yardage dredged is used in calculating the tenor. Nevertheless, the inclusion of tailings in royalty calculations has been contested by the head of the gold department of the Bolivian Mines Ministry.

When a dredge digs parallel courses there has to be a certain amount of re-dredging of its own tailings due to the arc the buckets make as they dig from top to bottom and to the fact that the tailings have a lower angle of repose than virgin ground. The amount of tailings to re-dig is small in percentage and varies with the conditions in any particular area. Sometimes it is possible to spread tailings away from the virgin ground; on other occasions it is necessary to stack them along virgin ground for water-level control.

The dredging of tailings has never yet affected the volume sufficiently to alter the sole tax scale, but the government agency has yet to approve officially any of the liquidations and the matter is of increasing concern.

A sliding-scale royalty and/or sole tax is a fair system for both the government and the operator of a property. If the ground is rich, the government gets a large percentage, but if it is low-grade the operator pays only a small percentage to the government so there is enough margin left to operate profitably.

As a result of the problem on tailings, the Ministry of Mines has refused lately to negotiate another contract with a sliding-scale sole tax based on tenor and has insisted on an increasing-scale basis. The new scale is not published yet, but it is understood that royalty plus sole tax starts at a combined 14 per cent of gross production and increases gradually to 30 per cent in 25 years.

DREDGE INSTALLATION

The cost of equipping a property in South America can be very high in an isolated location such as in Bolivia where it was necessary to construct airfields, buy aircraft of special type and fly everything in. Only $151 000 was spent in exploring the property in Bolivia before sufficient reserves were developed to justify the equipping of the property, which then cost almost four million dollars.

A second-hand dredge in Bulolo, New Guinea, which was originally designed for airlift, was purchased at the site, as it was, for $175 000. It cost $135 000 to dismantle, $40 000 to truck it to Lae, $70 000 to ship it from Lae to Arica, Chile, $57 000 to haul it from Arica to La Paz by rail, $25 000 to haul from La Paz to Caranavi by truck and $128 000 to fly it from Caranavi to Teoponte, the construction site. The last figure does not include the cost of two Northrop aircraft which amounted to $350 000.

The foregoing figures, except for the aircraft cost, are included in the dredge cost of $2 090 426, shown in Table V, which is a general breakdown of the costs of equipping the property in Bolivia with one 11-ft^3 dredge capable of digging 65 ft below water.

TABLE V

Dredge	$2 090 426
Diesel power plant	202 157
Transmission line	48 232
Buildings	385 227
Aircraft	385 455*
Airports	73 546
Transportation equipment	56 997
Office machines	6 126
Furniture and fixtures	18 979
Sundry equipment	29 282
Exploration equipment	29 091
Construction equipment	85 475
Mechanical shop equipment	89 554
Electrical equipment	7 800
Carpenter shop equipment	5 767
Hospital equipment	10 986
Materials and supplies	262 509
Exploration and development	151 083
	$3 938 692

*Two Northrop Airplanes	$349 079
One Helio Courier	36 376
	$385 455

More recently, a second-hand dredge was purchased by Pato for $50 000. Extensive modifications were done on this dredge to extend its digging depth from 65 to 90 ft. The hull was lengthened 24 ft, which required major modifications to the superstructure. The total cost of this dredge installed and in operation on the property in Colombia will approximate $1 807 654.

DREDGING

Tables VI–VIII provide operating data and general information on fifteen dredges used by the International Mining Corporation's associated companies in South America, while Tables IX–XI provide cost data on the twelve units which have been operating in recent years.

Owing to an extensive strike at Pato in 1963 cost figures for 1962 are used for that company as they are more representative of normal operation. Pato's dredge 6 has been shut down for several years because of a lack of reserves for this small specially designed dredge. Pato's dredge 8 started operation in August 1964, and Choco's dredge 2 was shut down the same month. Narino's dredge 1 has been shut down since early 1962 for lack of reserves. A study of these Tables readily indicates the difference in the character of the ground being dredged and the conditions under which the dredges are operating.

The Bolivian dredge—a single unit, supplied by air, operating in a remote area under the most difficult dredging conditions—is by far the poorest performer in capacity in relation to its size and has the highest monthly and unit costs. It was expected that performance would be poor and costs correspondingly high when the area was entered, but both have been worse than expected. The ground is very tight with many large boulders, most of them close to bedrock which makes it very difficult to clean bottom, and the wear on the bucket lips is hard to imagine. A 6-in welded lip segment will last only a few months as compared to a year and a half in the Pato and Choco operations in Colombia. As many as four lips have been worn off in a single year in Bolivia. The wear on all other parts of the dredge is also excessive in comparison to other dredges operating under less severe conditions. The ladder on the Bolivian dredge has been broken twice; the stacker has dropped three times; and the hull has been punctured twice. Screen plates and stacker belts have a very short life. The main dump hopper requires constant rebuilding, as does the ladder and practically every other operating part of the dredge.

Power costs of 6·1 cents/yd^3 in Bolivia and 2·3 cents/yd^3 in the Narino operation are a reflection not only of low yardage but also of the higher cost of diesel-generated power. At Choco, where some diesel power is used in conjuction with hydroelectric power, costs range from 0·5 to 1·0 cents/yd^3, which compares to costs of 0·2 to 0·3 cents at Pato where an efficient hydroelectric plant produces very low-cost power. In terms of cents/kWh, generating costs vary from 0·25 cents at Pato to 3·7 cents in Bolivia, with average costs of 0·62 cents/kWh for hydro and 2·1 cents/kWh for diesel at Choco and 2·6 cents/kWh for diesel at Narino.

Diesel fuel for the Bolivian operation is trucked down a precipitous road on the eastern slope of the Andes to the end of road transport. From there it is then floated in 1000-gal* rubber tanks approximately 40 miles down a swift river to the operating site. This results in diesel fuel costs of 33 cents per gallon landed at the operating site. In Narino

---

* U.S. gallon.

TABLE VI.—*General information and operating data on all dredges—Pato Consolidated Gold Dredging, Ltd.*

Dredge number and name:	1-A, Pato	2, Nechi	3, Jobo	4, San Fco.	5, Boyaca	6, Guay-abal	7-A, Sta. Margerita	8, Antioquía
Date of construction	1936	1915	1937	1938	1938	1950	1941	1936 (dwg.)
Last reconstruction	1947	1957	1953	1950	1958	—	1949	1964
Depth of hull, ft	7·75	9	10	11	11	7·5	9·1	11
Width of hull, ft	45	60	80·2	74	74	44	70	70·5
Length of hull, ft	105·3	125	157·4	170	165·3	92·5	166·05	173·25
Width of ladder well, ft	5·6	5·6	8·7	8·7	8·7	5	8	8·25
Flotation depth, ft	5	6·75	7·2	7·2	7·2	5·5	7·79	9
Digging face to end of stacker, ft	207·25	282	345·25	349·25	359·2	161·67	405·96	395
Top stern gantry to waterline, ft	58·25	69	77·67	77·67	77·67	44·21	93·5	72·3-Deck
Lower tumbler to end of tail sluices, ft	157·25	200·5	256·50	263·75	265·75	134·75	250·44	278·83
Length of digging ladder, ft	83·3	101	144	158	156	77·5	160·25	171·25
Max. digging depth, ft	45	48	81	91	87	39	87	90
Spuds	1	1	2	2	2	1	1	2
Size of buckets, ft^3	6	8·5	13·75	13·75	13·75	2·2	13·75	13·75
Pitch of buckets, in.	30	33¾	38	38	38	21	38	38
Number of buckets, average	71–75	71–75	87–100	105–109	105–109	92	105–109	117
Buckets per min	30	26	28	28·0	32	35	28·5	28
Size of revolving screen, ft	5·4 × 34·5	6 × 34·5	8 × 48	8 × 48	8 × 48	4 × 24	9·1 × 54	9·25 × 53·31 o.d.
Screen rev/min	10·50	10·7	8·25	8·25	8·25	13·4	7	6
Screen slope, in.	1–1½	1–1½	1–1½	1–1½	1–1½	1–1½	1–1½	1–1¾
Jigs	Yes	Yes	Yes	Yes	Yes	Yes	Yes	Yes
Theoretical maximum capacity, yd^3/month	288 000	353 520	615 600	604 800	703 641	122 400	615 600	615 600
Actual capacity, yd^3/month—recent years	145 000	224 000	400 000	477 000	491 000	51 000	329 000	Not known

TABLE VII.—*General information and operating data on all dredges—Choco*

	Dredge 2	Dredge 3	Dredge 4	Dredge 6
Date of construction	1920	1923	1932	1938
Last reconstruction	1949	1955	—	1957
Depth of hull, ft	8·5	10	9·75	9·75
Width of hull, ft	60	60	65	65
Length of hull, ft	110	120	120	126
Width of ladder well, ft	6	6	6·66	6·66
Flotation depth, ft	7·42	8·33	7·33	8
Digging face to end of stacker, ft	231·15	230	241·5	271·75
Top stern gantry to waterline, ft	62·75	64·5	55·5	55·5
Lower tumbler to end of tail sluices, ft	158·75	169	161	177·5
Length of digging ladder, ft	86	90	96·25	112·5
Max. digging depth, ft	34	42	42	54
Spuds	none	2	none	1
Size of buckets, ft^3	7	12	8	11
Pitch of buckets, in.	29	$33\frac{3}{4}$	$30\frac{1}{8}$	$31\frac{7}{16}$
Number of buckets, average	70	70	80	91
Buckets per min	24	22	34	32
Size of revolving screen, ft	6 × 32	7 × 36·06	7 × 36·5	7 × 36·5
Screen rev/min	9·8	8·85	8·3	7·67
Screen slope, in.	$1\frac{7}{8}$	$1\frac{3}{4}$	$1\frac{3}{4}$	$1\frac{3}{4}$
Jigs	No	No	No	Yes
Theoretical maximum capacity, yd^3/month	391 608	527 998	409 594	598 399
Actual capacity, yd^3/month—recent years	356 091	423 895	204 821	430 490

TABLE VIII.—*General information and operating data—Narino and Bolivia*

	Dredge 1	Dredge 5	Dredge 7
Date of construction	1915	1937	1934
Last reconstruction	1957	Proposed for 1965	1959
Depth of hull, ft	7	12·04	9·79
Width of hull, ft	45	70	68
Length of hull, ft	104	155	129·14
Width of ladder well, ft	5·5	6·625	6·75
Flotation depth, ft	5·5	9·04	7·79
Digging face to end of stacker, ft	212	306	296·5
Top stern gantry to waterline, ft	Not available	79	57
Lower tumbler to end of tail sluices, ft	" "	250 approx.	190
Length of digging ladder, ft	85·04	130	105
Max. digging depth, ft	35	66	54
Spuds	none	1	1
Size of buckets, ft^3	5	13·5	11
Pitch of buckets, in.	26	35	31¾
Number of buckets, average	87	95	88–94
Buckets per min	26	27	27
Size of revolving screen, ft	5·73 × 32·92	7 × 41·52	7·08 × 38·58
Screen rev/min	9·4	8·13	8·51
Screen slope, in.	1¾	1⅞	1¾
Jigs	No	No	Yes
Theoretical maximum capacity, yd^3/month	262 297	583 200	474 725
Actual capacity, yd^3/month—recent years	115 604	277 971	145 892

TABLE IX.—*1962 dredge statistics—Pato Consolidated Gold Dredging, Ltd.*

	Dredge 1-A		Dredge 2		Dredge 3		Dredge 4		Dredge 5		Dredge 7-A	
Bucket capacity, ft^3	6		8·5		13·5		13·5		13·5		13·5	
Yardage dug, average per month	115 250		199 916		366 750		494 500		529 333		351 500	
Costs, average per month	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3
	$		$		$		$		$		$	
Direct:												
Labour	3 786	3·3	3 936	2·0	4 037	1·1	4 178	0·9	4 377	0·8	3 930	1·1
Materials	1 595	1·4	1 504	0·7	4 479	1·2	7 422	1·5	5 844	1·1	4 886	1·4
Power	304	0·3	475	0·2	740	0·2	863	0·2	948	0·2	935	0·2
Production taxes and refining	308	0·3	362	0·2	669	0·2	1 608	0·3	1 047	0·2	858	0·2
Shop work	705	0·6	1 145	0·6	2 301	0·6	2 361	0·5	2 394	0·5	1 365	0·4
Insurance	203	0·2	203	0·1	710	0·2	710	0·1	710	0·1	710	0·2
Miscellaneous	1 936	1·6	1 790	0·9	1 706	0·5	2 421	0·5	2 196	0·4	3 007	0·9
	$ 8 837	7·7	$ 9 415	4·7	$14 642	4·0	$19 559	4·0	$17 516	3·3	$15 691	4·4
Indirect:												
Labour, materials and miscellaneous	8 611	7·5	8 611	4·3	8 611	2·3	8 611	1·7	8 611	1·6	8 611	2·4
Royalties	—		—		—		—		—		2 449	0·7
Depreciation	3 262	2·8	3 262	1·6	3 262	0·9	3 262	0·7	3 262	0·6	3 262	1·0
	$11 873	10·3	$11 873	5·9	$11 873	3·2	$11 873	2·4	$11 873	2·2	$14 322	4·1
Total costs	$20 710	18·0	$21 288	10·6	$26 515	7·2	$31 432	6·4	$29 389	5·5	$30 013	8·5

TABLE X.—*1963 dredge statistics—Choco group and Narino*

	CHOCO GROUP								NARINO	
	Dredge 2		Dredge 3		Dredge 4		Dredge 6		Dredge 5	
Bucket capacity, ft^3	7		12		8		11		12·7	
Yardage dug, average per month	340 214		480 981		259 577		525 102		277 970	
Costs, average per month	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3
	$		$		$		$		$	
Direct:										
Labour	7 311	2·1	7 681	1·6	6 162	2·4	7 952	1·5	5 555	2·0
Materials	6 376	1·9	8 932	1·9	6 036	2·3	8 347	1·6	6 880	2·5
Power	2 010	0·6	2 585	0·5	2 694	1·0	3 547	0·7	6 531	2·3
Production taxes and refining	142	—	561	0·1	653	0·2	153	—	364	0·1
Insurance	307	0·1	579	0·1	557	0·2	767	0·1	780	0·3
Amortization	—		—		—		4 314	0·8†	2 390	0·9*
Miscellaneous	260	0·1	686	0·2	685	0·3	467	0·1	1 501	0·5
	$16 406	4·8	$21 024	4·4	$16 787	6·4	$25 547	4·8	$24 001	8·6

TABLE X.—*1963 dredge statistics—Choco group and Narino (contd.)*

	CHOCO GROUP								NARINO	
	Dredge 2		Dredge 3		Dredge 4		Dredge 6		Dredge 5	
Bucket capacity, ft^3	7		12		8		11		12·7	
Yardage dug, average per month	340 214		480 981		259 577		525 102		277 970	
Costs, average per month	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3	Total	cents/ yd^3
	$		$		$		$		$	
Indirect:										
Labour	5 319	1·6	7 447	1·5	6 852	2·6	7 648	1·5	11 559	4·2
Materials	1 915	0·6	2 681	0·6	2 710	1·0	2 681	0·5	6 523	2·3
Royalties	873	0·3	—		4 798	1·9	828	0·2	2 597	0·9
Depreciation	806	0·2	1 128	0·2	967	0·4	1 128	0·2	1 272	0·5
Miscellaneous	2 132	0·6	2 985	0·6	3 784	1·5	3 300	0·6	5 505	2·0
	$11 045	3·3	$14 241	2·9	$19 111	7·4	$15 584	3·0	$27 456	9·9
Total costs	$27 451	8·1	$35 265	7·3	$35 898	13·8	$41 131	7·8	$51 457	18·5

* Amortization of deferred heavy spare parts. † Amortization of deferred dredge reconstruction.

TABLE XI.—*1963 dredge statistics—South American Placers, Inc.*

	Dredge 7	
Bucket capacity, ft^3	11	
Yardage dug, average per month	145 892	
Costs, average per month	*Total*	*cents/yd^3*
Direct:	$	
Labour	4 153	2·8
Materials	8 545	5·9
Power	8 887	6·1
Shop work	2 303	1·6
Laboratory and refining expenses	1 114	0·8
Insurance	1 169	0·8
Miscellaneous	2 064	1·4
	$28 235	19·4
Indirect:		
Labour, materials and miscellaneous	27 585	18·9
Royalties	8 279	5·7
Depreciation	26 864	18·4
	$62 728	43·0
Total costs	$90 963	62·4

and Choco the diesel fuel is transported up-river in company-owned barges and is landed at the operating sites at approximately 17 cents/gal.

Supplies for the Bolivian operation are flown from the end of road transport in company-owned tri-motored Northrop aircraft. These planes, which are not licensed in the U.S.A., were designed especially for short-field heavy-lift work in the Korean War, but were not completed before the end of that conflict. Two of these aircraft were purchased to haul the dredge to the operating site and they did the job well, hauling loads up to 5 tons and more from a short field built by the company at the road terminus. It now appears likely that a road will be constructed with U.S. AID (Agency for International Development) funds to mining and agricultural areas in the vicinity of our operation, and it is estimated that overall costs will thereby be lowered at least $80 000 per year with the elimination of the airlift and river transport of diesel fuel.

The Narino operation is in a narrow river valley with fairly hard digging in coarse material. The river fluctuates rapidly as much as 30 ft in twenty-four hours from the tropical rainfall which exceeds 300 in. per year. It has been a single-dredge operation except for 2 or 3 years when dredge 1 from Choco was moved down to dig some shallow areas.

Dredge 1 performed satisfactorily in the Choco, but poorly in the more compacted coarser alluvials of Narino. Direct costs are not too much higher than those at Choco, but the one dredge has to absorb all the indirect costs of supervision, shops, warehouses, commissary, hospital, schools, transport, housing, and all the other things that are essential to isolated operations.

The Choco operation has a variety of digging conditions. Dredge 4 is in a small swift river with coarse material and many boulders, although the digging is still easier than in Bolivia. Dredges 3 and 6 have been in a large inland area with relatively easy digging. For several years dredge 2 has been in an inland area where it has been able to handle large yardage for its size. All of the Choco dredges have had changes in bucket capacities, bucket-line speeds, and by other means operating performance has been generally improved.* The yardage has been increased as much as 286 per cent from the original performance of some of the dredges.

The Pato operation has the best overall digging conditions with easy digging and very few rocks larger than a man's head. The supply line from Barranquilla on the coast is relatively long, but river transport is available by private companies at reasonable rates. All of the dredges operate out of one main camp as compared to three camps in Choco and two in Narino. All transport to the dredges at Pato is by river launches. Power costs are low, as mentioned earlier. Indirect costs are distributed equally over six dredges and from July 1964 a seventh unit will help carry the overhead. The company has very modern and efficient shops, hospital, and other facilities. Pato also has extensive reserves, whereas Choco and Narino have very limited reserves.

Recovery Systems

Jigs are used as the primary recovery system on all the Pato dredges, on Choco's dredge 6, and on the Bolivian dredge. The other dredges have tables with standard riffles. The gold is so fine-grained at Pato that recoveries would be very poor without jigs. Choco's dredge 6 had jigs installed a few years ago and platinum recovery has definitely improved so jigs are soon to be installed on Choco's dredge 3.

Sales

The platinum produced by Choco is sold in the U.S.A. to refiners, generally through brokers. The gold produced in Bolivia is sold in the London market. Gold produced in Colombia is sold to the Bank of the Republic at $35 per oz with 25 per cent paid in dollars and 75 per cent in pesos at the average free rate of exchange during the previous week. Since August 1964 the Bank of the Republic has also been paying a bonus of 15 pesos per oz of gold to the large mining companies. A large bonus has been paid to small mines for several years and still continues.

---

* Romanowitz, C. M., High-speed bucket-lines. *Min. Wld, San Francisco*, **22**, July 1960, 39–43.

Future Outlook

Colombia has been experiencing an inflationary period during recent years, as have most other countries. In the past 11 or 12 years the peso has depreciated from 2.50 pesos per dollar to 13 pesos per dollar, which has been helpful to the mining industry because the currency was depreciating at a faster rate than costs were rising. However, costs continue to rise at an ever-increasing rate and the pressure for higher wages to offset increasing living costs is constant. Owing to an unfavourable balance of payments the importation of necessary equipment and spare parts is a very serious problem. Local manufacture of many items is increasing, but often, as with many new industries, the quality is not as good as it should be and the cost is generally considerably higher than that of the same imported item.

With a fixed-price product in an inflationary economy it has been necessary to steadily improve operating efficiencies and recoveries. Bucket-line speeds have been increased and jigs have been installed to improve recoveries that suffered with increased volumes. These improvements, plus some help from a depreciating currency and, more lately, the bonus in Colombia, have made it possible for the mining companies to continue on a profitable basis.

622.271.4:622.342.13(712.1)
338.58:622.342.13(712.1)

Paper 9

Operations of the Yukon Consolidated Gold Corporation, Canada

W. H. S. McFARLAND, B.Sc.(Min.)(*California*)
Consultant Engineer, The Yukon Consolidated Gold Corporation, Ltd., Canada

---

SYNOPSIS

Yukon Consolidated operates five dredges and one bulldozer-sluiceplate operation in the Klondike River area of the Yukon Territory, Canada, under sub-Arctic climatic conditions. Operations are restricted to the period between April and November. Dredges have been operated by this company or its predecessors continuously for 55 years.

Gold occurs partly in gravels and partly in deeply weathered bedrock. The gravels are overlain by a deposit of 'muck' of aeolian origin. All the superficial deposits are frozen perennially, and must be thawed before removal. The muck is thawed and removed by a combination of solar thaw and hydraulicking. Gravel and bedrock are thawed by circulated cold water which is introduced into the ground under pressure.

Prospecting and evaluation are carried out by churn drilling on a pattern. No casing is used in the frozen ground.

Present production is approximately 40 000–50 000 oz gold per annum from 4 000 000 yd^3 of material. Gold is coarse and is recovered on fibre mats in sluices. The dredges are driven by electricity generated in the Company's own plant by water power.

The Yukon Consolidated Gold Corporation, Ltd., is a Canadian company carrying on operations for the recovery of gold from the placer deposits of the Klondike goldfields in the Yukon Territory. Its properties lie in the valley of the Klondike River and its tributaries, Bonanza, Eldorado, and Hunker Creeks and in the valley of Indian River and its tributaries, Dominion Creek, Sulphur Creek, Gold Run Creek, and Quartz Creek. These cover a belt approximately 20 miles wide by 50 miles long. At the end of 1938 they consisted of 1650 placer mining claims and 9 leases or concessions. By the end of 1963 much of this area had been mined out, so that the properties then consisted of 813 placer mining claims and 2 concessions.

Dawson, the operating headquarters of operations, is situated at the confluence of the Klondike and Yukon Rivers, about 46 miles east of the Yukon–Alaska border at latitude 46°05′ north and longitude 139°20′ west, about 250 miles south of the Arctic Circle. Climatic conditions approximate those of the Arctic regions, winters being long and severe but summers warm. From the middle of May to the first part of August there is no night.

GRAVELS WORKED

Gold-bearing gravels of the Klondike District are of three kinds:

(1) *the Hill or 'White Channel' gravels* occupying the ancient stream beds, called 'White Channel' gravels because of the large content of nearly pure quartz; they lie from 200 ft to over 300 ft above the present stream beds and vary in thickness to over 200 ft. Their gold values are mostly near bedrock. These gravels were drift-mined by the early miners and later worked by hydraulicking by the Yukon Gold Company;

(2) *the Bench gravels*, which occupy terraces left by the streams in cutting down to their present levels; they are composed of quartz pebbles and boulders mixed with fragments of the underlying schist, sand and silt, and in quantity are of minor importance, but on Dominion Creek some of them have proved to be very highly profitable when worked by bulldozers pushing the gravels into sluice boxes for the gold recovery; and

(3) *the Creek gravels* covering the beds of the present streams; their composition being about the same as that of the Bench gravels. These gravels were of the first importance as they were to contain all the dredging activities except in the one case of dredge no. 12—a very small dredge of $2\frac{1}{2}$-ft^3 bucket capacity—purchased to mine certain Bench gravels on Dominion Creek.

MINING TENURE

Properties in the Yukon are not held in freehold but in the form of mining claims and mining leases granted by the Dominion Government under the provisions of the Yukon Placer Mining Act. Full-size creek mining claims are 500 ft along the valley by 1000 ft on each side of the base line or from the centre of the valley if no base line has been surveyed. Full-size hill and bench claims are 500 ft along the valley by 1000 ft at right angles to this direction. All claims are held on the basis of an annual renewal fee of $10 for each claim plus representation work of an assessed value of $200 per claim per annum. Leases are held under the terms set out in the lease when it was granted, which terms provide for certain fees to be paid and the amount of representation work to be performed each year. For the purpose of such representation work claims may be grouped so that the work performed on one claim may be applied to other claims. It is only in this way that it becomes possible to hold enough ground for large-scale operations over a period of years.

RECENT DEVELOPMENT

There were three main reasons for the development of the Yukon dredging industry in recent years: (1) the development of a successful method for stripping the frozen muck which overlies the gold-bearing gravels; (2) the development of a method for thawing the frozen gravels with water at natural temperatures instead of, as in the early days, by steam; and (3) the increased price of gold.

In the early days of the Yukon dredging industry, when thawing by steam was practised and both muck and gravel were thawed and dredged, it cost between 35 cents and 40 cents per cubic yard to dredge ground which was over 50 per cent naturally thawed. With the introduction of new methods for stripping and thawing, these costs were reduced to 16 to 20 cents per cubic yard dredged in ground which was practically 100 per cent frozen and the unit value of each cubic yard dredged was increased by every cubic yard of barren muck removed.

ASSESSMENT OF CLAIMS

In 1934 much of the company's holdings had not been tested for gold values. The lease on the Klondike River had been proved by actual dredging with large dredges to contain sufficient gold values to warrant dredging where the gravels were naturally thawed. On the smaller streams, which had been worked over by hand methods in the early days of the district, it had been demonstrated by the operations of the Yukon Gold Company and the New Northwest Corporation that any ground which had been successfully hand-mined would warrant dredging, but the actual values remaining in the 'old works' and the extent and value of any workable ground adjacent to the 'old works' could only be estimated.

A four-year campaign of prospect drilling was initiated under expert supervision to provide reliable figures for estimating yardage and value in each dredging area and it was on the basis of these figures that equipment was ordered.

The technique of placer drilling in frozen ground with Keystone drills had been developed in Alaska to a point of high accuracy. Frozen ground simplifies many drilling problems. No casing is required and, properly handled, it reduces sloughing to a minimum and enables accurate measurements to be made of the volume of the material extracted from the hole by adding measured quantities of water and determining the rise in the hole by tape and float. The arrangement and spacing of the drill-holes varies according to conditions, but the bulk of the Company's reserves were estimated from drilling done on cross-section lines 500 ft apart. Depending on the width of the paystreak holes on the cross-section lines were spaced from 50 to 300 ft apart. At Dawson when a new area is to be tested scout drilling is first done with the holes so located that they will fit into any closer drilling which may be done at a later time. The drilled cross-section lines are studied for depths of muck, dredging section (gravel plus pay bedrock) and gold content. These calculations are made by connecting up the holes of one cross-section line to the holes of the next cross-section line in a series of triangles and estimating the volume and gold content of each triangle by weighting the holes at the apices.

By the end of 1937 the drilling campaign had established proved gravel reserves of 96 360 000 yd³ containing $42 154 600 (gold at $35) with an estimated profit, after allowing for necessary capital expenditure, of $22 780 000. Of the 96 360 000 yd³ proved, approximately 5 000 000 yd³ represented Hill gravels on Hunker Creek which were worked by hydraulicking and 3 000 000 yd³ represented Bench gravels on Dominion Creek, which are now being worked by bulldozer and sluice box and by dredge no. 12.

Ground being worked

The ground which has been and is now being worked by the company falls into two general classes:

(1) ground worked over by the early day miners and early day mining companies ('old works');

(2) virgin ground which was of too low a grade to work by earlier methods.

Approximately one-third of the company's proved reserves were in virgin ground and the balance in 'old works'.

OPERATIONS

Fig. 1 shows the location of operations.

Stripping

In the Dawson district, in the smaller streams, the gravels are overlain with a deposit of barren, frozen muck ranging from 10 to 65 ft in depth. This is a very difficult and expensive material to thaw and dredge. It can be removed by hydraulic stripping at a lower cost than by thawing and dredging, and every yard of muck removed increases the unit value of each yard dredged and enables more bedrock to be worked over in a season. Working a dredging area on the creeks means clearing the surface of brush, trees, old buildings and machinery and then removing as much of the muck as available runoff grade will permit.

When operations were first started in 1934 the clearing of brush and trees was carried on by hand work under contract, but as tractors came into use this work was shifted to bulldozers.

Water supply for stripping is derived from ditches or from pumps taking water from the local streams. Pumps for this purpose were standardized on a high-efficiency 10-in centrifugal pump rated at 3000 U.S. gal/min against a 150-ft head. Each unit is driven by a 150-hp

2300-V synchronous motor—the capacity of which is just right to supply a 3½-in nozzle. The pump casings were made heavy enough to stand double pressure so that two pumps could be operated in series if higher pressure were required, and the pump set-ups were made so that operation could be switched from parallel to series as required. Each pumping unit and its starting equipment is housed in a portable house 18 ft long by 10 ft wide built on four steel-shod skids which are double-ended so that the building can be shifted in either direction. The building has a 3-in floor nailed directly to the skids and a 2-in by 4-in framework covered with corrugated iron; there are fourteen of these units.

Stripping is accomplished by hydraulicking, using water under pressure through a system of pipes leading to a series of 'No. 2 Giants', placed so that all surrounding muck can be reached with a minimum shifting of Giants. To get the best efficiency out of the water it is necessary to cover a large area in a single operation so as to give the sun's rays time to thaw the surface. For each stream of water used from eight to ten Giants are set up, so that by a process of rotation from one to another sufficient time elapses after all thawed material has been removed from a given set-up before it is necessary to work it over a second time. At the peak of operations there were approximately 140 Giants set up at one time over all the stripping plants.

Water duties at Dawson vary from 8 yd^3 to 15 yd^3 per 'M.I.D.'* depending on the character of the material to be removed. Stripping is complicated by the presence of 'old works' tailings on the surface, deep moss, and slide rock in the muck. For efficient work pressures between 50 and 120 lb/in^2 are desirable.

The style of pipeline layout for placing the Giants and for delivery of water to them is determined by the shape of the area to be worked, the slope of the surface and the presence of tailings on the surface. Originally, the first set-up of the Giants was for the purpose of disposing of the tailings but, after bulldozers became available this work was done mechanically. The distance between Giants is regulated by the water pressure and the character of the material to be handled. A rule-of-thumb method for placing Giants for muck stripping is to work away from each Giant a distance in feet equal to one-and-a-half times the pressure of water in pounds per square inch.

Sulphur–Australia ditch

A Sulphur–Australia ditch was constructed to supply water for Sulphur Creek, which has a very small supply of its own. Fortunately,

---

* M.I., Miner's Inch, a flow of 1½ ft^3 of water per minute.

M.I.D., Miner's Inch Day, flow of 1 Miner's Inch for 1 day, i.e. 1½ ft^3/min for 24 hours.

Australia Creek was near and had a much more satisfactory water run-off. This creek was dammed off at a point about 6 miles from its mouth and the water was led into a 1500-M.I. capacity ditch which was constructed to carry the water to a point on Dominion Creek, near the mouth of Sulphur Creek, where it was carried across the valley of Dominion Creek in a pipeline salvaged from the operations of the Yukon Gold Company. A surge tank was built at the end of the pipeline and from this the water was led into two centrifugal pumps, each driven by 800-hp motors, and these elevated the water some 300 ft to another 1500-M.I. capacity ditch which delivered it up to Sulphur Creek; this ditch was 11 miles long. At the end of this ditch another pump station was constructed to elevate a smaller part of the water another some 300 ft to supply upper Sulphur Creek operations.

It is interesting to note how surge suppression was managed at the two stations. At the main pump station each pump was equipped with a heavy flywheel to keep the pumps revolving for a few seconds after any sudden power failure. In addition, each pump outlet was supplied with a gate valve and a swing-check valve. The swing-check valve was equipped with a dashpot arrangement which would prevent that valve from closing too quickly. The flywheels gave a sufficient time interval

Example 1

15 feet of muck removed by stripping

Annual capacity of dredge	650 000 yd³
Stripping cost	6 cents/yd³
Thawing cost	4 cents/yd³
Dredging cost	10 cents/yd³

Working cost per ft² of bedrock

Stripping 15 ft³ of muck @ 6¢ per yd³		3·3¢	
Thawing 15 ft³ of D.S. @ 4¢ per yd³		2·2¢	
Dredging 15 ft³ of D.S. @ 10¢ per yd³		5·5¢	
	Total	11·0¢	
Working cost per yd³ dredged		19·8¢	
Value per yd³ dredged		45·0¢	
Profit per yd³ dredged	Total	25·2¢	
Ft² of bedrock worked over per season			1 170 000
Production per season			$292 500
Working cost per season			128 700
Working profit per season		Total	$163 800

to allow for this closing. The gate valve was supplied to enable the pumps to be started against a closed valve and to regulate the water supply to the pumps when the head was less than full supply.

At the smaller station on Sulphur Creek surge was handled by erecting an air cylinder on the pipeline beyond the gate valve and swing-check valves. On the smaller station this worked very well.

Stripping before thawing and dredging

As the muck is very difficult to thaw, the removal of as much as possible greatly reduced the cost of thawing the remaining material. The examples illustrate the importance of removing the barren muck prior to dredging. Assumed figures approximate average conditions in the dredging reserves and cost figures approximate those at the time when operations were being expanded. Depth of muck is assumed to be 20 ft, gravel 6 ft, pay bedrock 4 ft; value per square ft of bedrock is 25 cents.

These examples show an increase of working profit produced by stripping of $105 950 per season. These figures were conservative because the thawing of muck, being difficult, is not often accomplished at 100 per cent efficiency, thus increasing the cost of dredging and decreasing the capacity of the dredge.

Example 2

No stripping done

Yd^3 dredged equal to 90 per cent of yd^3 thawed

Annual capacity of dredge	650 000 yd^3
Thawing cost	5·5 cents/yd^3
Dredging cost	10·0 cents/yd^3

Working cost per ft^2 of bedrock

Thawing 30 ft^3 @ 5·5¢ per yd^3		6·1¢
Dredging 27 ft^3 @ 10·0¢ per yd^3		10·0¢
	Total	16·1¢
Working cost per yd^3 dredged		16·1¢
Value per yd^3 dredged		25·0¢
Profit per yd^3 dredged	Total	8·9¢

Ft^2 of bedrock worked over per season		650 000
Production per season		$162 500
Working cost per season		104 650
Working profit per season	Total	$57 850

Over the years, with increasing costs of labour and supplies, the costs of stripping, thawing and dredging have necessarily advanced. Beyond endeavouring to increase the efficiency of the work not much has been changed in the stripping and dredging operations, but in the thawing operation the advance in costs has been checked to a certain extent by changing the spacing of the points. Experience has demonstrated that in some areas the original spacing of points on 16-ft triangles could be increased to as much as 24 ft, thus reducing the number of distributing pipes to be laid and greatly decreasing the number of points to be driven. Also, with the wider spacing of points, enough equipment could be placed before actual thawing was begun to cover an entire season's operation and no equipment needed to be shifted during the season. With ground conditions varying a great deal in different areas the wider spacing could not be universally adopted, but the distributing pipes were laid out for this spacing and any extra points required were given water from hoses leading from multiple cross-heads in the header pipes. At the end of 1963 stripping, thawing and dredging costs then averaged as follows:

Stripping per yd³	10·8¢
Thawing per yd³ in stripped areas	7·2¢
Thawing per yd³ where the formation is largely muck	11·1¢
Dredging per yd³	20·1¢

Applying these figures to the conditions shown in Example 1 gives the following results:

Example 1 (*modified*)

15 ft of muck removed

Working cost per ft² of bedrock

Stripping 15 ft³ of muck @ 10·8¢ per yd³		6·0¢
Thawing 15 ft³ of muck, gravel and bedrock @ 7·2¢ per yd³		4·0¢
Dredging 15 ft³ @ 20·1¢ per yd³		11·2¢
	Total	21·2¢
Working cost per yd³ dredged		38·2¢
Value per yd³ dredged		45·0¢
Profit per yd³ dredged		6·8¢
Ft² of bedrock worked over per season		1 170 000
Production per season		$292 500
Working cost per season		248 300
Working profit per season		$44 200

Example 2 (*modified*)

No stripping done

Thawing cost per yd^3		11·1¢
Dredging cost per yd^3		20·1¢
Yd^3 dredged equal to 90 per cent of yd^3 thawed		

Working cost per ft^2 of bedrock

Thawing 30 ft^3 @ 11·1¢		12·3¢
Dredging 27 ft^3 @ 20·1¢		20·1¢
	Total	32·4¢
Working cost per yd^3 dredged		32·4¢
Value per yd^3 dredged		25·0¢
Working profit (or loss) per yd^3 dredged		(7·4¢)
Ft^2 of bedrock worked over per season		650 000
Production per season		$162 500
Working cost per season		210 600
Working profit (or loss) per season		($48 100)

These examples show that even with the increase of cost stripping off the barren muck is very advantageous.

Thawing

Thawing with water at natural temperatures follows completion of stripping and the more complete the job of stripping the fewer are the thawing problems. Thawing is accomplished by injecting water into the ground through pipes termed ‘points’ which are driven by hand to bedrock as the water thaws the ground ahead of them. This system was started by Miles in Alaska and was perfected at Nome, Fairbanks, and Dawson for large-scale operations. The equipment now used at Dawson is mostly the same as that which was perfected at Nome and Fairbanks by the United States Smelting Refining and Mining Company.

Originally, the ground to be thawed was laid off in a series of blocks, or units, each containing about 600 points, but with the wider spacing of points this was increased in size. Now, with the wider spacing of the ‘header pipes’, enough equipment is laid to cover a full season’s area without shifting.

Water is led through a 12-in gate valve into a flanged pipe ranging in size from 11 in. down to 8 in. and having 6-in outlets spaced at distances to suit the location of the ‘header pipes’. These outlets have 6-in gate valves and to these are attached the 'header pipes' which distribute the water to the points. Header pipes are 6-in and 4-in slip joint pipes which are 16-ft long when driven. Each piece of header pipe has a 2-in screwed outlet in the centre for the 'cross-heads', which consist of a 2-in by 4-in nipple leading into a 1-in by 1-in by 2-in tee. From each side of the tee is a 1-in by 4-in nipple, a 1-in cock and a 1-in Chicago Pneumatic hose coupling, thus giving a lead to two points at each cross-head. Connection to the points when they are spaced 16 ft apart is by 1-in hose 16 ft long, which has a Chicago Pneumatic hose coupling on one end and a $\frac{3}{4}$-in gooseneck on the other end. When point spacing is greater than 16 ft the length of the hose is made longer to fit.

'Points' consist of $\frac{3}{4}$-in and $\frac{1}{2}$-in extra-heavy pipe, the former having a special chisel-bit on the end; it is used where there is considerable driving to be done. The bit is a combination of steel which is drop-forged so there is high-carbon steel on the cutting edge and mild steel on the end which is welded to the pipe. The $\frac{1}{2}$-in pipe has no tip and is used where driving is a secondary consideration and ease of cleaning plugged pipes is primary. The use of $\frac{3}{4}$-in points is predominant. Points are in 10-ft lengths and are coupled together with hydraulic recessed couplings. If additional outlets are required at the cross-heads this can be accomplished by screwing the standard 2-outlet cross-head into a 2-in tee or a 2-in cross as the case may be.

Points are driven to bedrock by men known as 'point drivers', using a driving weight slotted to fit around the point and held in place by a $\frac{3}{8}$-in round-steel bar fitted through holes in the weight. This also acts as a handle. The driving weight is brought into contact with an anvil in the shape of a heavy swing-bolt clamp tightened on the point. A wooden handle bolted to the back of the clamp acts as the means of twisting the point. Twisting of the point is essential to keep it going straight when being driven. Both weight and clamp are drop-forgings of special manufacture. After the points have reached bedrock they are taken over by 'point-doctors', who keep them operating until thawing of the ground is completed.

At Dawson the duty of the water varies between 5 and 10 yd^3 thawed per M.I.D. of water, point driving rates ranging from 2 to 15 ft per man-hour. The thawing season extends from about 10th May to 25th September, preparatory work beginning about 15th April. After the thawing season is over work continues to about 20th October on the dismantling of the equipment and the preparation of the pump site for the following season. Owing to the scarcity of supply, water for thawing has to be recirculated. Settling basins are built to settle out the mud, and water from these basins is pumped to the thawing field, from where it flows back to the settling basin. All water passing to the pumps must first pass through mechanical screens for removal of floating trash. These screens have baskets which are covered with 7-mesh wire screen galvanized after weaving.

Pumps for circulating thawing water were standardized on a 12-in centrifugal unit rated at 6000 U.S. gal/min against a 100-ft head and driven by a 200-hp 2300-V synchronous motor. These pumps have a flat performance curve so that they can be operated at a lower head with a corresponding increase in quantity of water delivered without a great loss of efficiency. Each unit, with its starting equipment, is installed in a portable house of the same dimensions and construction as used for the stripping pumps; there are eleven such units.

It is interesting to note that under present climatic conditions ground once well thawed improves rather than having any tendency to freeze back. Naturally, the surface freezes during the winter to a depth of 4–6 ft, but this frost usually goes by early June. Early snowfalls help to keep this freezing to a minimum.

Dredging

Dredging of the ground follows completion of the thawing. Dredging in the Yukon is relatively good when the ground is well thawed but, as that is a condition which is not always easy or practicable to attain, the dredges are often called upon to dig a certain amount of frozen ground.

The dredging season is relatively short and major repairs are made in the shutdown period as far as possible. Preparatory work begins about 1st April; the dredges start operating about 1st May, continue to the early part of November and the crews are generally laid off within two weeks of the time dredging is closed down.

Hydraulic mining

Hydraulic mining of the Bench gravels of Hunker Creek was carried out on Paradise Hill located on the left limit of Hester Creek at its junction with Hunker Creek. This deposit was ideally located for hydraulic mining except for the water supply, the only available source being Hunker Creek which, by itself, was inadequate. To remedy this condition it was decided to construct a settling basin and recirculate the water. As dredge no. 11 had already mined out the creek gravels below this area, there was no interference with dredging operations.

A large dam of dredge tailings was built about a mile below the place where tailings from Paradise Hill were to be dumped into Hunker Creek. Below this dam another low dam was built to give a supply to a pump station. The company had on hand a 1200-hp centrifugal 4-stage pump capable of delivering 500 M.I. against a 600-ft head. This pump and its electrical equipment were reconditioned and placed in service. A system for surge suppression, similar to that supplied to the Upper

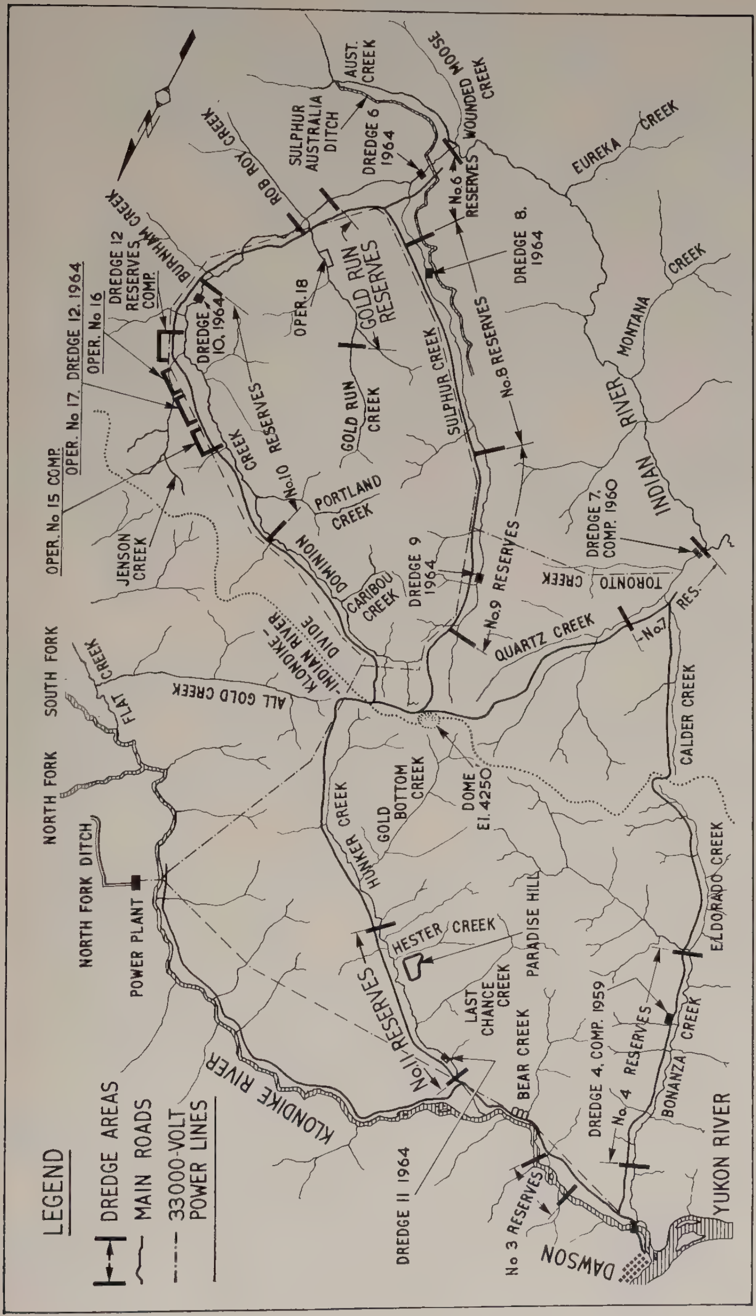


Fig. 1.—Location of operations.

Sulphur pump station, was supplied here. In operation the large tailing dam was tight enough to back up the water and make a settling basin, but seeped enough water to supply the pump station below it without its taking any dirty water direct. The recirculation of water worked so well that even in very dry periods hydraulicking was carried on at full force.

The bedrock on Paradise Hill was extremely uneven, sometimes varying as much as 50 ft in a distance of little more than 100 ft, but generally had a good slope towards Hunker Creek so that no bedrock cuts were required. The bottom gravels were pushed to the head of the sluice boxes by bulldozer and the cleaning of the bedrock was performed by the same method. As with the bulldozer operations described below, the sluice boxes had angle-iron riffles over which perforated plates were placed in a step manner to prevent the riffles clogging with heavy concentrates.

Hydraulicking was started in 1952 and completed in 1960. There was no operation during 1959 while the tailings dam was being rebuilt after it had been largely destroyed during a flood at the beginning of the season.

Bulldozing

Certain areas which are too small or too shallow or have very uneven bedrock are being mined by the 'bulldozer' method. In this operation the start is the same as for a dredging area. The surface is first cleared of old buildings, brush and trees and then the muck is stripped off by hydraulicking. Generally the gravel is shallow enough to thaw naturally after the muck is removed and no thawing is required if this condition holds true.

When the deposit is ready for the gold recovery a sluice box with angle-iron riffles is set up with a wide dump box at its head into which a bulldozer pushes the gravels. There is a suitable washing arrangement over this dump box so that the material is well washed before it enters the riffle area in the sluice. Boulders are generally removed from the dump box before they can enter the sluice. It has been found good practice to place perforated plates over the riffles, in a stepped arrangement, to prevent the riffles from clogging with heavy concentrates and leave them free to hold the gold.

Where there is not sufficient grade below the sluice box to allow the tailings to run away from the box it is necessary to have the bulldozer stack them as often as required.

The company operates a large, well equipped repair shop at its main camp situated at Bear Creek on the Klondike River about 8 miles from the city of Dawson. Here it also maintains a large stock of repair parts and general supplies.

TABLE I.—*Operating data: stripping—thawing—dredging*

Year	Cubic yards of muck stripped	Stripping cost		Cubic yards of D.S. thawed	Thawing cost		Cubic yards dredged (all dredges)	Total dredging costs (including stripping and thawing write-offs)		Production (all dredges)		
		Total $	Cents per yd³		Total $	Cents per yd³		Total $	Cents per yd³	Total $	Cents per yd³	Per cent of estimate
1935	416 438	62 615·37	15·04	958 824	88 237·00	9·20	5 222 144	555 590·13	10·64	908 339·04	17·39	106·0*
1936	1 290 220	128 122·76	9·93	2 046 748	106 775·09	5·21	7 957 108	876 448·80	11·02	1 456 423·77	18·31	107·0*
1937	1 871 624	170 898·78	9·13	3 844 638	188 068·27	4·89	7 443 785	842 001·56	11·31	1 887 723·05	17·30	115·2
1938	2 818 268	201 621·47	7·15	4 140 821	263 864·38	6·37	8 550 652	1 174 081·59	13·73	2 127 103·43	24·88	111·9
1939	3 762 824	187 864·74	4·99	4 112 845	242 081·70	5·89	10 090 182	1 285 322·08	12·74	2 734 778·15	27·10	110·3
1940	4 298 047	229 530·61	5·34	6 174 169	290 149·53	4·70	10 480 799	1 197 409·28	11·43	2 617 227·40	24·97	102·3
1941	2 692 219	197 829·53	7·35	7 108 325	311 739·38	4·39	8 205 270	1 163 342·10	14·17	2 333 681·40	28·43	108·6
1942	2 501 681	174 284·51	6·97	4 529 117	202 647·88	4·47	10 401 131	1 478 142·29	14·21	2 831 292·56	27·22	112·0
1943	360 621	31 861·00	8·84	486 021	12 016·39	2·47	7 006 055	823 723·09	11·76	1 329 354·64	18·98	98·1
1944	1 016 994	84 565·21	8·40	—	—	—	4 348 874	596 932·78	13·73	616 288·73	14·17	105·5
1945	1 038 110	96 923·82	9·33	477 056	38 032·08	7·97	2 682 743	584 758·71	21·79	915 962·91	34·14	136·9
1946	2 138 654	204 385·39	9·54	2 868 542	185 256·27	6·46	5 614 700	983 737·76	17·52	1 362 697·63	24·27	101·3
1947	2 290 715	295 613·45	12·91	5 048 008	340 268·99	6·74	6 692 528	1 167 938·49	17·45	1 326 906·80	19·82	133·7
1948	3 144 020	261 167·02	8·31	4 415 705	324 632·71	7·35	7 146 666	1 355 269·92	18·97	1 652 795·72	23·13	110·3
1949	2 487 131	219 609·69	8·83	3 850 978	222 212·84	5·77	8 556 300	1 456 644·02	17·02	2 171 909·31	25·38	85·3
1950	2 037 891	223 906·13	10·99	4 155 557	247 855·45	5·96	6 835 964	1 466 837·93	21·46	2 540 841·38	37·17	142·8
1951	1 736 120	201 632·75	11·61	4 946 601	232 678·51	4·70	5 878 422	1 305 260·72	22·20	1 932 525·93	32·87	131·7
1952	1 982 404	254 837·97	12·85	4 249 900	209 754·89	4·94	5 863 491	1 431 740·20	24·42	1 900 880·41	32·42	112·8
1953	1 937 568	243 005·47	12·54	4 562 819	237 211·29	5·20	4 340 557	1 256 678·00	28·95	1 416 883·07	32·64	104·5
1954	1 728 937	263 910·68	15·26	3 042 778	163 316·03	5·37	5 115 815	1 307 680·10	25·56	1 780 387·84	34·80	104·5
1955	2 238 528	248 054·85	11·08	3 171 720	162 946·76	5·14	5 319 022	1 295 861·35	24·36	1 524 058·08	28·63	99·8
1956	1 438 987	207 428·01	14·41	2 241 394	116 017·38	5·18	4 960 352	1 240 631·14	25·01	1 604 861·99	32·35	119·1
1957	1 242 555	164 384·59	13·23	2 376 147	128 162·90	5·39	6 283 046	1 426 830·62	22·71	1 822 579·94	29·01	101·1
1958	1 042 469	129 643·62	12·40	3 546 725	170 681·54	4·81	6 130 347	1 414 845·32	23·10	1 671 801·57	27·30	93·2
1959	1 296 134	155 735·57	12·01	2 149 194	138 748·16	6·46	5 914 587	1 541 870·28	26·10	1 724 037·04	29·20	87·9
1960	1 020 689	122 660·00	12·01	3 456 012	234 659·00	6·79	4 517 964	1 443 021·00	31·90	1 993 864·00	44·10	125·3
1961	1 214 439	108 013·00	8·89	2 545 886	221 281·00	8·69	4 041 022	1 267 591·00	31·30	1 554 118·00	38·40	114·1
1962	1 015 621	108 645·00	10·70	3 876 533	280 484·00	7·20	4 477 386	1 247 670·00	27·90	1 299 684·00	29·10	129·8
1963	1 431 141	148 338·00	10·36	2 656 277	226 264·00	8·52	4 478 831	1 409 049·00	31·50	1 275 247·00	28·50	87·2
Totals	53 491 049	5 127 088·99	9·58	97 039 340	5 586 043·42	5·76	184 555 743	34 596 909·26	18·75	50 314 254·79	27·26	110·0

* Percentage of actual value recovered over estimated recovery is approximate for 1935 and 1936.

Camps

Originally each operation had its own camp, but in later years it was found more economical to maintain fewer camps and give the men transportation to and from the job. Camps are now maintained at Bear Creek, Middle Dominion Creek, Granville, Upper Sulphur Creek and the Power Plant.

Operating data

Table I gives figures relating to the work done in preparing the ground and subsequent dredging, from 1935 to 1963, showing:
(1) cubic yards of overburden removed by stripping and the cost;
(2) cubic yards of dredging section thawed and cost;
(3) cubic yards mined by the dredges and the cost; and
(4) the recovery from the ground by the dredges and the percentage of value recovered compared with estimate of value made on the basis of the prospect drilling in prospected areas, and from other information where prospect drilling results were not available.

Tables II and III summarize by years the results of the hydraulicking operation on Paradise Hill and the bulldozing operation on the Dominion Benches.

TABLE II.—*Operating data, Paradise Hill—no. 13—hydraulicking*

Year	Duty of water	Cubic yards mined	Cost		Production—Au at $35		
			Total $	Cents per yd^{3}	Total $	Cents per yd^{3}	Per cent of estimate
1952	7·24	111 651	38 100·65	34·12	16 847·31	15·09	54·9
1953	—	127 445	52 523·41	41·21	106 013·75	83·18	277·3
1954	4·80	186 026	102 078·94	54·87	73 685·76	39·61	133·9
1955	4·81	164 587	111 295·45	67·62	175 010·38	106·33	191·0
1956	4·11	177 208	102 070·92	57·60	76 591·25	43·22	80·4
1957	8·27	371 570	105 519·68	28·40	111 738·40	30·07	87·8
1958	7·25	325 936	86 393·09	26·50	115 960·69	35·60	118·6
1960	4·26	169 370	116 045·00	68·50	92 229·00	54·50	136·1
Totals		1 633 793	714 027·14	43·70	768 076·54	47·01	173·1

TABLE III.—*Bulldozing operating data, Dominion benches 14, 15, 16*

Year	Cubic yards mined	Cost		Production—Au at $35		
		Total $	Cents per yd^{3}	Total $	Cents per yd^{3}	Per cent of estimate
1958	153 987	105 729·36	68·7	104 968·19	68·1	38·2
1959	120 650	101 633·85	84·2	70 011·41	58·0	80·7
1960	138 763	150 845·00	108·7	79 734·00	57·5	65·7
1961	257 192	161 690·00	62·9	351 254·00	136·6	106·1
1962	205 537	153 497·00	74·7	212 827·00	103·5	89·2
1963	264 890	169 126·00	63·8	231 106·00	87·2	112·0
Totals	1 141 019	842 521·21	73·8	1 049 900·60	92·0	78·0

Hydroelectric power plant

All dredges were electrically operated with power supplied by the company's hydroelectric plant situated on the Klondike River. As originally constructed this had two 5000-hp Francis type, I.P. Morris turbine units supplied by water taken from the North Fork of the Klondike River. This was delivered to the plant through a 6-mile canal which gave a head of about 220 ft. When plans were made to extend the company's operations more power was required and another 5000-hp I.P. Morris turbine unit was added. As the North Fork River had periods when its water supply was well below the required amount, a 16-mile canal of 10 000 M.I. capacity was constructed to bring water from the South Fork of the Klondike River and deliver it to the North Fork above the intake of the original canal. At the same time the capacity of the original ditch was increased to take care of the greater water supply. This was done by using a dragline shovel along the upper bank of the canal. The canal is now 50 ft wide on the water line, carries an average depth of water of 6 ft and has a capacity of 30 000 M.I.

Power is delivered to the dredges, to Bear Creek (the main operating headquarters) and to the city of Dawson over 33 000-V lines. Secondary lines are 2300-V.

In the fall of the year, when cold weather comes, power is erratic during the freeze-up. This condition continues until the canals and the rivers freeze over—after which no further trouble is experienced from slush ice. As the ice in the canal gets thicker the water delivered gradually falls off, but the output of the plant seldom gets below 2500 kW, which is sufficient for all winter uses. A short, sharp freeze and an early snowfall give the ideal conditions for the freeze-up.

The shut-off gates at the intakes of the canals are housed over and kept warm with electric heat so that the gates can always be operated as required.

Freightage

Freight rates between Vancouver and Dawson vary between $55 and $135 per ton. To this must be added the cost of local transportation to destination which, in the past, averaged about 50 cents per ton-mile but, as road conditions were improved, has been considerably reduced and is now about half that amount.

Discussion

(Papers 5–9)

Mr. J. E. Denyer,* introducing Papers 5 to 9, said that they dealt with the recovery of metals from alluvial deposits. Although the contribution to the world's supply of all metallic raw materials made by alluvial mining was comparatively small, it was nevertheless important and, in the case of tin, vitally important, for about 70 per cent of the Free World's tin came from alluvial deposits.

Mr. J. K. Broadhurst and Mr. D. J. Batzer, in their paper 'Valuation of alluvial tin deposits in Malaya, with special reference to exploitation by dredging', described methods which could, in part, be applied to deposits containing other metals. Accurate valuation of alluvial deposits was of vital importance when the expenditure of large amounts of capital was contemplated—a modern dredge cost well over one million pounds—and inaccuracies which might arise unless great care were taken could have very expensive consequences. The drilling of bore-holes in alluvial deposits was cheap and quick, but unless proper precautions were exercised in dealing with the material brought to the surface the results could be misleading. The first problem was to determine the volume of material *in situ* represented by what was brought to the surface. Errors in determining that would cause corresponding errors in the resulting valuation, and the authors confessed that the main uncertainty in valuation concerned the measurement of core volume. (That point was also stressed in Paper 8.) On the other hand, Mr. W. H. S. McFarland drew attention to the fact that in the frozen gravels of the Yukon the measurement of bore-hole volume presented no problem. Broadhurst and Batzer described the methods used and have established that to allow for the lower compaction of tailings, as compared with virgin ground, a factor of 80 per cent must be applied.

The next step was the determination of the tin content of the bore-hole samples. The paper described how that was done by small-scale gravity concentration methods, including the use of the *dulang* for the first stages and other means for the final stages. There were two reasons for the use of such complicated methods instead of chemical assays:

---

*Siamese Tin Syndicate, Ltd., London.

in the first place the tin content of the ground was so small—usually less than 0·05 per cent—that chemical assays were unreliable; and, secondly, a chemical assay would report all the tin present, including small particles of cassiterite attached to larger particles of barren material and also very fine particles of cassiterite, neither of which could be caught in the treatment plant.

Originally, it was the practice to determine the average value of an area as a whole, by taking the average value of all the bore-holes. The statistical method described by the authors should result in a more reliable aggregate figure, possibly with a smaller number of bore-holes, provided that their warning that it should not be applied or interpreted mechanically was heeded. The wide variations that had been found in practice between individual bore-hole values and dredge recovery were vividly illustrated in Fig. 2 (p. 110), although it appeared that the average was in the region of 100 per cent. Now that the richest alluvial tin deposits had been exhausted, and future production must come to an increasing extent from low-grade deposits which were very near the borderline of profitability, accurate valuation was of great importance and in that respect the paper was particularly timely.

Two papers dealt in some detail with orthodox dredging, but under widely differing climatic conditions: Mr. Patrick H. O'Neill's entitled 'Gold and platinum dredging in Colombia and Bolivia, South America', and Mr. W. H. S. McFarland's on 'Operations of the Yukon Consolidated Gold Corporation, Canada'. Formidable natural difficulties had been successfully overcome at the sites described in both papers; in South America the lack of roads necessitated buying special types of aircraft and building airfields so that everything could be brought in by air, and in northern Canada the arctic winters could bring work to a halt for four or five months each year. The Yukon Consolidated Gold Corporation and its predecessors had been operating for 55 years, but the development of the industry in recent years had been brought about largely by two technical improvements—a successful method of stripping frozen overburden and the use of cold water instead of steam for thawing the gold-bearing gravels. Those methods, which were described in detail in the paper, had reduced working costs very substantially and, by the removal of barren overburden, the gold content of the material dredged had been almost doubled.

Mr. O'Neill's paper described the equipment used in South America for drilling in circumstances which differed widely from place to place, and also something of the difficulties which mining companies might encounter in dealing with governments and landowners both before and after starting operations. It was evident that operating conditions were in many respects extremely adverse because of the high cost of transport and the tropical rainfall, which could cause great and rapid fluctuations of the water-level in rivers. In some cases very hard digging led to high maintenance costs.

Papers 8 and 9 were to a great extent factual, containing a wealth of information about the plant used, and were particularly valuable in that they gave, in considerable detail, costs of the various operations and of equipping mines.

Hydraulic mining of alluvial deposits was described by Mr. S. Pakianathan and Mr. P. Simpson in their paper 'General practices on large-scale hydraulic mines in Malaysia'. (There was also mention of hydraulic mining, albeit on a limited scale, in Paper 9.) In Canada the topography allowed the sand and gravel to be washed to the sluice-boxes, where the gold was recovered without intermediate pumping. Also, bulldozers were used in conjunction with monitors—in fact, in some of the smaller deposits the excavation and transport were done entirely by bulldozers. In Malaysia conditions were somewhat different. Many of the hydraulic mines were situated so that they could get water throughout the year at sufficient pressure from the nearby mountain ranges, but the sluiced tin-bearing sand and gravel could not flow away by gravity and must be elevated by gravel pumps. Paper 6 described the method of concentration, by means of *palongs* and jigs, and dealt with the problem of disposing of the large quantities of tailings. While a dredge stacked its tailings neatly and tidily behind it, that was not the case with a hydraulic mine and various methods, some of which added considerably to the cost of mining, had to be used, particularly if rehabilitation of the mined areas was compulsory.

Such a method of mining was generally more expensive than dredging and consequently the ground must be somewhat richer, but hydraulic mining could be used for deposits which, because of their small size and uneven contours, were unsuitable for dredging. One great advantage of hydraulicking was that deep depressions and crevices in the bedrock, which might contain high concentrations of cassiterite, could be excavated without difficulty. Dredges could not dig into those crevices and it had been suggested that tin-bearing areas, either virgin or tailings, with a highly uneven bedrock, which would otherwise be dredged, might be worked by hydraulic methods. Although the paper described the operations at large hydraulic mines, it should not be overlooked that in Malaysia there were many small mines, mostly owned by Chinese, whose methods were basically the same, and that together the hydraulic mines accounted for about 40 per cent of Malaysian tin production.

'Mining offshore alluvials' by Mr. M. J. Cruickshank differed slightly from others in Session 2. They gave interesting and valuable information about what had or was happening in alluvial mining, whereas Paper 7 pointed to the future. As had been stated in the Paley report,* world reserves of minerals were limited and there was every prospect of an acute shortage of essential raw materials towards the end of the twentieth century, and a more recent paper† repeated that story. The words and figures were different, but the message was the same—something had to be done! Mr. Cruickshank gave grounds for hope and he not only described what could be done, but *how* it can be done. The Continental Shelf, as yet almost unexplored, had an area of about ten million square miles or 15–20 per cent of the total land surface. Much of it was accessible for mining and there was no reason to suppose that it did not contain mineral deposits similar to those on land. The difficulties of mining were greater, but not insuperable. Drilling for oil, not merely offshore but in the middle of the North Sea, was now a practical possibility, and boring for tin deposits off the coasts of Siam, Malaysia and Indonesia was commonplace. Dredging with bucket and grab dredges for offshore tin deposits had been practised successfully for some years in the East and Mr. Cruickshank described the use of airlift dredges for recovering diamonds off the stormy coast of South West Africa. Numerous other instances of submarine alluvial mining for material ranging from sand and gravel to gold were also cited. The Appendix showed that a very significant start had been made towards exploiting the riches of the Continental Shelf, but it was obviously only a beginning.

Mr. I. R. M. Chaston,‡ commenting on Paper 5, said that although he had spent several years in Malaya working on the problems of the tin-mining industry, he still found some points in the paper which were not clear. In the section headed 'Tailings' (pp. 100–2) it was said that, taken at their face value, the values of bores in tailings indicated dredge recoveries of the order of 65–75 per cent. What followed, however, suggested that it was not the face value of the bore results which was considered but a much more complex figure related to the total tin content of the ground. By 'face value' one would assume that the average bore value of the virgin ground was compared to the average value of the tailings. Thus if the measured value of the feed were 1 kati/yd^3, and the measured value of the tailings were 0·5 kati/yd^3, then on face value one would say that the dredge recovery was 50 per cent. From that simple viewpoint the expansion of the ground after treatment should have tended to result in an overestimation of dredge recovery, rather than an underestimation. In the example given on page 101 the true assay of the feed excavated was 1 kati/yd^3 and the true assay of the tailings was 0·83 kati/yd^3. That suggested that there had been a recovery of 17 per cent of the tin present when, in fact, there had been more. Normally, therefore, it could be held that the compacted tailings value would give a more reliable recovery figure than the adjusted one.

---

* Paley, W. S. *Resources for freedom*, Vols. I–V (Washington: U.S. Gov. Printing Office, 1952), 819 p.

† Dunn, J. A. Fundamental metal and mineral problems. *Min. chem. Engng Rev.*, **56**, no. 10, July, 1964, 28–32.

‡ Cornish Tin Smelting Co., Ltd., Cornwall.

The authors had written (p. 101): 'A certain plant loss is inevitable and probably amounts to between 10 and 15 per cent'. Perhaps they could expand on that and say to what that loss referred. In one of the few published papers* to give detailed dredge sampling data the results showed that although the feed to the dredge sampled was in no way exceptional, and contained neither a great deal of clay nor a great deal of fine cassiterite, the actual recovery of tin was below 50 per cent. It was emphasized that that figure, being based on chemical assays, was the recovery of the tin in the feed.

The results of that dredge sampling supported the present authors in showing that, on the dredge, no −300-mesh cassiterite was recovered by the jigs. However, other results quoted in the paper showed that, with correct feed preparation, it was possible to recover a substantial part of the −300-mesh cassiterite with jigs.

It therefore seemed that what the authors and others meant when they said that the dredge recovery was 85–90 per cent was of the form: 'Of the cassiterite in the ground which it could conservatively be thought possible to recover in the adverse conditions under which it was chosen to operate the treatment plant, 85–90 per cent will actually be recovered'.

In the speaker's opinion it would lead to much faster development in improvements in the recovery plant if that recovery estimate were rephrased in the form: 'The loss of tin on the dredge was, say, 50 per cent, of which, say, 20 per cent was lost in fines which it had not been possible to recover, 15 per cent was lost in locked particles and 15 per cent had been lost in the course of processing'.

Expressing dredge recovery in those terms might bring home to those responsible for the direction and financing of dredge operations the opportunities which existed for improvement. The dredge might then cease to be a digging machine with a recovery unit tacked on the back and become a properly integrated mining and treatment plant.

Continuing through the paper, the authors were asked to explain the heading to the middle section of Table III (p. 103). Should that

---

*Chaston, I. R. M. Developments in the treatment of Malayan tin ores. *International mineral processing congress 1960* (London: Institution of Mining and Metallurgy, 1960), 593–609.

have read *Free recoverable cassiterite in the tailings from treatment of the barren overburden sample*? If so, could the authors give the actual amounts of concentrate which were recovered from washing and panning those so-called barren overburden samples?

The assay certificate reproduced in Table IV (p. 104) showed how the assay adjustment factor was arrived at. In the example given the amount of tin concentrate recovered by treatment of the *amang* was small, with the result that the assay adjustment factor was low at 77 per cent. Line (*d*) in Table IV showed that the tin in the *amang* was equivalent to 1·0 g of 72 per cent Sn concentrate, of which an amount equivalent to 0·47 g of 72 per cent Sn was recovered. In other words, the recovery of tin from the *amang* was only 47 per cent. In the example shown that loss would have very little effect on the total bore assay, but where the assay adjustment factor was high the proportion of the reported tin content which had been recovered from the *amang* must also have been high. If the recovery of tin from the *amang* were equally low in those cases, then the loss would have a considerable effect on the bore value. It would be interesting if the authors could show the assay certificates for the bores *S* 15 and *AC* 39 of Table III where the assay adjustment factor had been high. From the figures given it was estimated that in those two bores nearly half the tin finally reported must have come from the re-treatment of the *amang*.

Although, as the speaker suggested, it was possible to recover fine, −300-mesh cassiterite in the primary jigging operation it was, in his and others' experience, easy to lose those hard-won values in the cleaner jigging stages. When the necessary improvements had been made to the primary jigging operation modification of the cleaning procedure would then be required. The speaker's suggestion was that it would be necessary to screen out the relatively very small amount of fines from the primary jig concentrate either before or after the cleaner jigging and treat those fines on shaking tables either ashore or on a separate, more stable, pontoon next to the dredge.

Turning to Paper 8 by Mr. O'Neill, he said that in describing the difficulties of airlifting the dredge into Bolivia the author had omitted one detail which had come to his attention on a recent visit to Bolivia. In flying into Teoponte one did not fly over the mountains, one flew through them! In constant cloud the pilots flew through the winding ravines with a stopwatch and a compass—so many seconds in one direction followed by so many in another. The hazard must add considerably to the cost of the operation.

He wondered if the depreciation figures quoted in the paper were not rather conservative. The sale value of the dredge in Bolivia must be considered negligible compared to the cost incurred in installation and therefore the entire cost of over $3 000 000 had to be written off during the working life of the dredge. The figure of $26 864 per month for depreciation, given in Table XI (p. 177), suggested that the write-off period envisaged by the company was of the order of 10 years. In the disturbed political conditions existing in South America generally, and indeed in many other parts of the world as well, did the author agree that in estimating for a project of that nature a shorter write-off period should be budgeted for? The depreciation figures for the other dredges were all very low; did that mean that all those dredges had been operating long enough to write off the initial capitalization?

Mr. J. S. Sheppard, O.B.E.,* referring to Paper 7, said there was very little published regarding offshore mining and it was not easy to obtain reliable information regarding such operations. The paper would be used as a source of reference by those interested in the working of offshore alluvials.

The list of current operations on pages 150–2 made no mention of the very extensive winning of sand and gravel being undertaken in the seas around Britain. It was true that on page 146 the author mentioned dredging in the English Channel, but the data supplied were incomplete and in part misleading. The author gave the impression that dredging for gravel was a comparatively new enterprise and was confined to the English Channel. Large tonnages had, in fact, been won for over twenty years from several areas around the coast. The estimated production for 1964 was of the following order:

	million tons
Mersey and S. Irish Sea	1
Bristol Channel	2
Off Isle of Wight	$1\frac{3}{4}$
Southern North Sea	$1\frac{1}{4}$

The author's mention of 26 000 square miles of accessible sand and gravel deposits in the English Channel would make operators smile wryly, for after many years of search they had found less than six square miles of suitable material that could be safely and profitably worked. It was doubtful whether the Continental Shelf Act would stimulate activity. Operators had been licensed to dredge in areas up to 12 miles out at sea for many years. It was the growing demand for the product that would be the stimulant.

A growing shortage of gravel for use as concrete aggregate in Southern

---

*Royal School of Mines, London.

England was causing some concern and many companies had searched for suitable offshore material within reach of the London market. Dredging companies in the Thames Estuary were prepared to dredge in 120 ft of water and to make a round trip of up to 150 miles. The modern suction dredges employed cost about one-third of a million pounds to build and carried from 2000 to 3000 t of material.

It was possible that by 1968 the tonnage of gravel won from offshore deposits around Britain would have risen to 10 000 000 t per annum. That quantity would have a cash value ex-wharf of, say, $30 000 000.

An operation of such a size and of such value to the community deserved a place in the table of operations; indeed, it could be described as being at or near to the top of the league table!

Mr. I. H. McLean* said that he would welcome Mr. Cruickshank's account of his own or anyone else's experience on the efficiency of a drop corer as a tool to retain sand and gravel samples from the sea bed.

Currently they had learnt of the complications of anchoring a 500-t drilling ship by six anchors. It would be of interest to learn whether 'dynamic positioning' could obviate entirely the necessity for such anchoring even in fairly troubled water.

In the operation at Nome, Alaska (pp. 140–1), the $44 000 quoted was presumably for the geophysical survey only. The cost of the subsequent drilling would be of great interest.

Mr. D. W. Saunders† said that having been intimately concerned as joint patentee over a number of years in the development of one of the dredges described in Paper 7, he would like to elaborate on the offshore aspects of the Dinosaur automatic grab dredge. Fig. 4 (p. 135) was, in fact, one of the earliest conceptions for paddock working—not for offshore alluvial mining. Believing that the future lay more in offshore dredging they had since that time concentrated on the development of a fully automated piece of dredging equipment of high output which would work in any condition of sea irrespective of depth.

To achieve that they had had to enter an entirely new field, that of electronics, and had introduced a system of grab control whereby the grab holding and closing drums were mechanically independent, thereby not only shortening the cycle time involved but also enabling the dredge to work, if necessary, in an ocean swell of say 50 ft, with the grab staying on the bottom during the closing part of the cycle instead of being plucked out partly closed.

---

*Hoveringham Gravels, Ltd., Newark, Nottinghamshire.

†Sir Bruce White, Wolfe Barry and Partners (Consulting Engineers), London.

For offshore work they would not propose the swivelling arrangement shown in the illustration, but would mount the unit as a fixture at deck level with the dredged material being conveyed from a sump under the dumping hopper by gravel pumps back to the plant. The dredging pattern would be maintained by movement of the entire dredge by means of the usual mooring wires, the winches for which would be integrated to the grabbing cycle.

Possibly an even more important consideration in offshore dredging was the ability to know exactly where one was in relation to the shore. For that a position-fixing system developed by The Decca Navigator Co., Ltd., had been adopted. Known as the 'Hi-Fix', it was particularly suited to the 'Dinosaur' as the receiving aerial could be fixed in a position vertically above the grab. With that system the path of the grab was plotted together with spot soundings by a stylus on a chart so that accurate plans of the operation could be kept and the dredge-master could see exactly what and where he was dredging.

It was also possible to return to the exact position and resume dredging at any time to an accuracy of ± 4 ft at a range of 100 miles.

To complete the picture, he would like to mention one further refinement upon which they were working and which was of particular interest in offshore work. That concerned the linking of the 'Hi-Fix' co-ordinates via a small computer to a pair of Voith-Schneider propellers whereby the dredge could be held exactly in position or moved in any desired direction irrespective of wind or currents. The accuracy they hoped to achieve was ± 2 ft at a range of 30–40 miles, thus dispensing entirely with moorings of any sort.

Mr. J. A'C. Bergne* said that if the provisional figures given on page 126 proved correct the density of manganese nodules was of the order of 8 lb/yd^2. His calculations were that with that density a trawl 10 yd wide at 4 m.p.h. could gather about 250 t/hour. Thus one could envisage future sea drifting for minerals as well as for food. The percentage recovery and percentage dilution factors for such an operation, however, had yet to be determined.

Mr. Cruickshank gave general details of position-fixing offshore and in mid-ocean. It might be interesting if he mentioned in addition the offshore method of horizontal sextant angles from which the position of an exploration craft could be plotted by any competent untrained man within ±10 yd if a chart were prepared giving arcs of equal angle circumscribed about each of a series of, say, three or four pairs of prominent landmarks. Such a chart could be drawn by any competent surveyor and resembled the common 'Deccanav' charts in the way it was used. He himself had constructed and used such a chart with arcs at 2½° intervals.

---

*Independent Consultant.

Again, the interpretation of 'sparker' records for the Channel tunnel survey made a fine picture in Fig. 1 (p. 128). The 'sparker' had also provided spectacular results in other parts of the world in geological structure interpretations. However good that tool was for that particular phase of exploration, there still remained the unavoidable need for probing the sea bottom to obtain samples of the actual material sought.

Coming to the use of sampling equipment, the manually operated Banka-type drill was still used in Malaya and he had personal experience of its use on rafts in rivers and in tidal waters. Alternatively, there were light power-operated casing rotating sampling machines which could also be used in such conditions of steady rise and fall of the raft in relation to the collar of the hole being drilled. The use of such gear under unprotected, offshore conditions, however, was severely limited by the degree of swell encountered. Even a 5-ft swell, almost invisible from the land in calm conditions, was enough to render the addition of lengths of casing hazardous and to give rise to a high risk of damage to casing once embedded in a hole.

Alternatively, drop coring was successful for geological sampling, but could not penetrate more than a few feet into either soft rock or sand. The Mackereth Sampler was, however, very successful in taking deep cores in soft muds.

With regard to large (24 in. in diameter) rotated casing systems, those were of necessity heavy and cumbersome and inevitably required craft of considerable size to carry them. That, in turn, increased the cost of evaluation greatly. A successful light and cheap system had yet to be developed to obtain true cored samples for depths of over 10 ft of underwater sediments, consolidated or unconsolidated, in unprotected offshore waters. Until such a system came to light proper evaluation of offshore deposits would remain costly.

Mr. M. A. Brooke* asked Mr. Cruickshank for information regarding the statement made on page 143 with reference to the Collins operation: 'Some difficulties have been reported to date in cleaning to bedrock'. He would be very interested to hear what those difficulties were and how the organization discovered losses on bedrock.

---

* Department of National Development, Australian High Commission, London.

He thought that the value of the operation for items 4 and 5 (Table II, p. 150) had been seriously underestimated. In 1963 Aokam Tin, Ltd., recovered 850 t of tin concentrate, 960 t in 1962 and 378 t for the first half of 1964. That would put the valuation of their output from, say, £500 000 to £750 000. Likewise the recovery by the sea dredge of Tongkah Harbour in 1962 and 1963 was 1840 and 1820 t respectively, with 1030 t for the first nine months of 1964. That would put the value of their output even higher—say from £1 000 000 to £1 500 000.

Fig. A was supplied by Mr. T. E. Miles of Adelaide, South Australia, a grandson of the late Captain E. T. Miles, who spent many years in the dredging industry in Thailand.

The print appeared in the *Journal of the Royal Thai Department of Mines* (date unknown). The dredge, which was certainly working in the Harbour at Bhuket in 1907–08, was the first dredge to work for tin in the Far East and, as such, was the ancestor of all the dredges that had ever worked in Thailand, Malaysia, Burma and Indonesia. Moreover, as it was working in the sea it must have been one of the earliest mining operations offshore.

The design of the plant was particularly interesting. It was copied from gold dredges working in Australia with which Captain Miles (a Tasmanian) was familiar. No details of the manufacturer, Simons & Co., or of the consulting engineer, Lewis A. Smart, were available.

The more interesting points about the dredge were:

(1) It had no protection from the rain—not necessary in Australia, but no doubt soon found essential in Bhuket where the rainfall was of the order of 150 in. per annum, falling in the short monsoon season.

(2) The screen oversize was dumped well behind by a conveyor belt stacker boom arrangement—again no doubt very essential in Australia when dredging for gold, but not necessary at Bhuket.

(3) The cassiterite-saving sluice-boxes ran athwart-ships and were very short. Losses of cassiterite, particularly of the fines, must have been considerable.

The successor company, Tongkah Harbour Tin, Ltd., re-dredged the tailings from the first two or three dredges, obtaining reasonable returns from that operation.

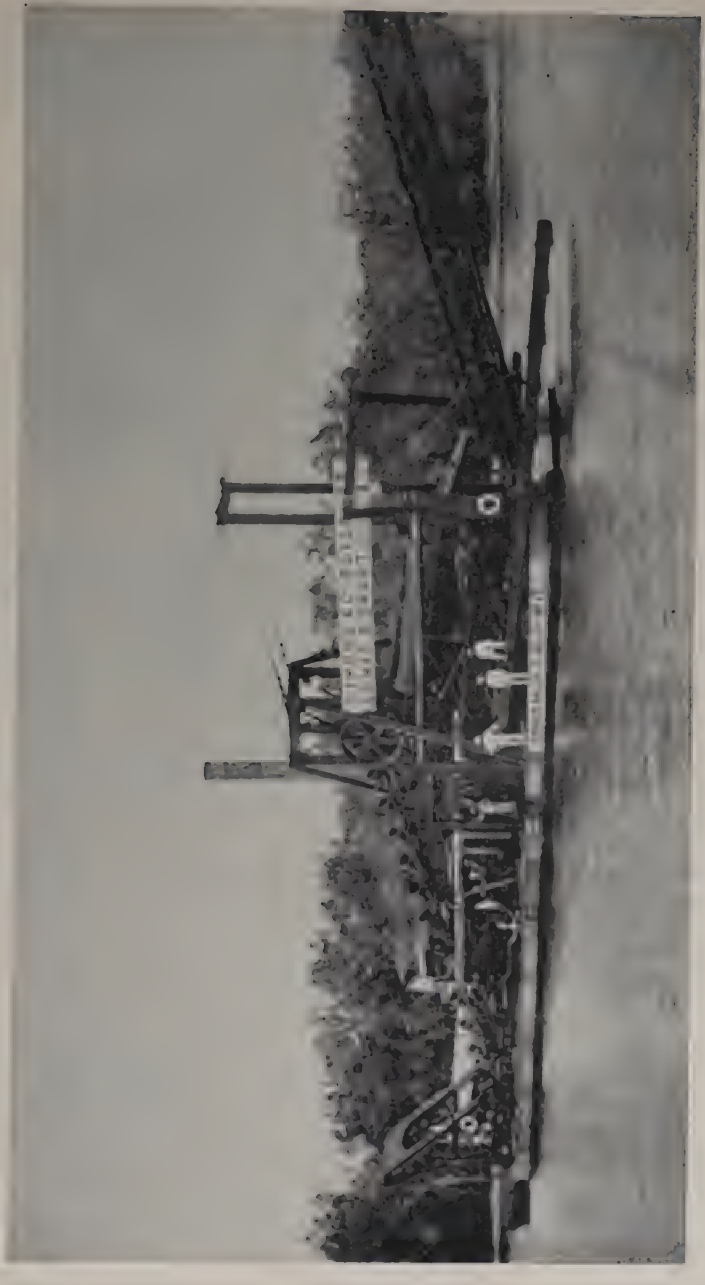


Fig. A.—First tin dredge operating in the Far East.

Mr. C. J. Simpson* said that in Tables VI–VIII of Paper 8 (pp. 171–3) dredge throughput was given as theoretical and actual, ranging from 30 to 90 per cent on various dredges. Could the author also give running time per month and percentage bucket capacity allowed when calculating the actual yardage?

Mr. R. F. Lethbridge† said the papers considered had described the various methods of alluvial mining in current use. He hoped it might be indicated why suction cutter methods did not seem to have been adopted in the great bucket-dredging and other alluvial areas of the world. He understood that such methods had been tried many years ago and were not successful; Paper 3 by Mr. J. C. C. Hill seemed to show that improvement had been made and one might well think that under proper control even in bad bottom conditions there could possibly be an improvement by the stripping of overburden first and then suction dredging to clean up between pinnacles.

Mr. J. C. C. Hill‡ said that in the working of dry opencast mines detailed consideration was given to the selection, from the various types of earthmoving plant available, of the most suitable equipment for the deposit. Normally, a combination of plant was applied; thus large bucket-wheel excavators were used in conjunction with the smaller face shovels and draglines, and the scrapers and bulldozers in conjunction with hydraulic monitors and pumping.

In dredging operations, however, it would appear that in most cases the tendency was to apply only the large bucket dredge, even though a proportion—sometimes a considerable proportion—of the mineral-bearing ground was left behind.

It was not proposed that suction-dredging equipment should replace the bucket dredge, but in certain areas it would appear that there was considerable scope for alternative equipment to be used in conjunction with the bucket dredge in order to achieve a better recovery.

In connection with the clean-up of bedrock, one could only speak in general terms. What could be carried out in the dry could be done underwater, although the cost was higher. It was therefore a question of economics as to whether the modifications to the bucket-dredge operation and the additional cost of secondary clean-up equipment and personnel would result in a payable recovery that would normally be left behind and covered with tailings.

---

*Tronoh Mines, Ltd., London.

†Rio Tinto-Zinc Corporation, Ltd., London.

‡Independent Consultant, London.

Underwater prospecting for river and offshore alluvials had been successfully carried out on one property using a 5- to 6-ft diameter casing in which the excavation was undertaken by an airlift controlled by a diver. That method was used in 60 ft of water and sampling was carried out through 12 ft of gravel and sand to bedrock.

Mr. J. A'C. Bergne said that because Malayan conditions included stiff clay, frequent thin rich washes on bedrock and uneven hard bedrock the suction cutter failed because: (*a*) the stiff clay clogged the cutter apertures; (*b*) the rotary cutters, being less robust than the standard bucket lips, failed in hard or pinnacle bottom; (*c*) on any bottom the clean-up, with rotary cutters, was not as thorough as with buckets; (*d*) in pinnacle bottom the stiff suction pipe could manoeuvre no more easily between pinnacles than a bucket ladder and, being less robust, was not so good at forcibly removing pinnacles, large blocks of which were too large to pass the cutter apertures; (*e*) logs and other large obstructions were impossible to move with suction cutters; (*f*) the mechanical transmission of power to the cutter head in early designs of suction cutters was by a rotating shaft mounted on the suction pipe and, as such, was highly vulnerable; and (*g*) the lateral strength of the rising main was insufficient to withstand side pressures encountered when traversing a face.

[Mr. Bergne then drew an illustration of pinnacle ground (caused by differential solution of limestone along intersecting series of fractures) and showed how he thought a suction cutter dredge with an articulated rising main could attack pinnacle bottom and clean-up successfully by maintaining a vertical aspect of the cutter.]

WRITTEN CONTRIBUTIONS

Mr. C. J. A. van Lummel:* In reading Paper 5, 'Valuation of alluvial tin deposits in Malaya', I was somewhat disappointed not to find more about the possibilities of determining tin contents by means of the spectrograph, X-ray fluorescence or even X-ray diffraction. I do not believe that 'physical or chemical locking' would present difficulties to these kinds of analyses, but I can imagine that their sensitivity is still somewhat low for the purpose. Would the authors consider it advantageous to pursue the matter, if only to have a check on the *dulang* method and for controlling concentration processes?

---

*Technical University of Delft, Holland.

Has any tin content ever been detected in the quartz, feldspars and micas of the Malayan alluvial deposits? In 1897 a geological study on the tin islands of Banka and Billiton appeared.* On pages 129–135 of the article the pioneer geologist Dr. Verbeek describes the way in which several analyses were performed on the above-mentioned minerals, gathered by him on the islands. The results showed tin contents of 0·01 to 0·03 per cent, presumably not in the form of cassiterite. In 1948 I had a check analysis made on Banka and found a tin content of more than 0·01 per cent for a carefully cleaned fragment from the interior of a big quartz crystal. Existing background tin content of the minerals accompanying tinstone would doubtless complicate matters.

Mr. G. M. Beaton and Mr. C. J. Simpson:† Our observations are confined to the information and data submitted in Paper 7 regarding tin-mining operations off the coasts of Malaya and Thailand.

With regard to Fig. 2 (p. 131) and Tables I and II (pp. 149–52), the following corrections are made to the facts given by the author: There are three dredges working off the coast of West Thailand, one grab dredge (1957) and one bucket dredge (1964) owned by Aokam Tin, Ltd., and one bucket dredge (1960) owned by Tongkah Harbour Tin Dredging, Ltd.

Grab dredge—The Aokam Tin, Ltd., grab dredge has two 4-yd^3 (nominal) grabs. The dredge is designed with winches capable of accommodating ropes to allow working to 200 ft, but at no time have ropes been fitted to reach to more than 125–130 ft. This was estimated to be the maximum depth that would be called for.

Dredging depth—The Aokam Tin, Ltd., bucket dredge is capable of reaching 116–120 ft below the water-level, not 60–70 ft as quoted. The Tongkah Harbour dredge reaches to around 100 ft, not 155 ft as stated. It should be noted that the deepest digging tin dredge is the Ayer Hitam Tin Dredging, Ltd., No. 2 dredge—a paddock dredge working in Malaya capable of digging to 153 ft below the water-level. Additional dredging depth can be achieved by lowering the level of

---

*Verbeek, R. D. M. Geological observations of Banka and Billiton islands. *Jaarb. Mijnw. Ned. Oost-Indië*, 1897, 272 p. (Dutch text.)

†Tronoh Mines, Ltd., London.

the dredge paddock by pumping—obviously not possible at sea. The author implies in Fig. 2 that a sea-working ladder dredge to reach a depth of 250 ft below the water-level is contemplated. The engineering problems and capital cost of such a dredge would be enormous and would require very high recoveries to justify them.

The capacities given in Fig. 2 for bucket and grab dredges require explanation. For the 14-ft^3 bucket dredges now working offshore in Thailand a throughput of 300 yd^3/h is a possible target. The author's upper limit figure of 3000 yd^3/h—representing, for a 600-h month, 1 800 000 yd^3—is in our view impossible to achieve. No operating dredge has regularly been credited with more than 500 000 yd^3 per month under the easier conditions of paddock dredging where a large proportion of the ground dredged is barren overburden and is removed at a rate greater than is possible for ore-bearing material requiring treatment. Several schemes for the economical removal of barren overburden have been considered; one now being explored is for a dredge with two bucket chains digging overburden and pay-dirt simultaneously. It has been suggested that a dredge of 18–20 ft^3 capacity might handle 1 000 000 yd^3 per month at about 6d./yd^3.

For the Aokam Tin, Ltd., grab dredge, also working offshore in Thailand, a monthly yardage of 120 000 yd^3 is considered acceptable—which represents 200 yd^3/h assuming a 600-hour working month. The figure of 15 000 yd^3/h given by the author as a proposed target needs to be explained.

In recent years paddock dredges have improved on the running times, mainly due to more planned maintenance of more reliable plant. Dredges working at sea do not encounter as good conditions as paddock dredges; uncertain weather will reduce running time; and shallower deposits result in reduced depths of digging face. The wave motion of the open sea also affects the 'bite' and hence the digging capacity of the bucket. In this connection the idea of a dredge with a pontoon able to submerge below the surface wave motion and with treatment plant mounted on a platform above is suggested.

The facts given in the section 'Tin-bearing sands' and in Table II regarding Aokam Tin, Ltd., in the area west of the Malay peninsula are inaccurate. The Company has no plans to build a bucket dredge for working there. The dredge shown in Fig. 9 (p. 141) is a grab dredge.

Mr. P. W. Dobson:* I would like to record one or two remarks which may be of interest to those members concerned with dredging techniques. In particular, these pertain to Papers 2, 3 and 7 and are prompted chiefly by the stimulating informal discussions on alluvial dredging.

---

*Head Wrightson Teesdale, Ltd., Yorkshire.

Speaking for the Company which is concerned with the development and manufacture of the 'Dinosaur' grab dredge referred to in Mr. Cruickshank's paper, it is encouraging to note that many aspects of this work have been tackled in a similar manner in Germany. It is clear that the concept of a fully automatic grab dredge is a practical proposition, although it is accepted that in most cases an operator is still required on the machine in primary dredging operations to deal with any large rocks or timbers, etc., brought up in the grab. Overriding manual controls are, of course, provided for this purpose.

The mechanical separation of the holding and closing rope hauling systems, and the provision of electrical speed control to these, facilitates the attainment of short overall cycle times without compromising between speeds of grab closing, hoisting and lowering. At the same time load-sharing during the hoisting operation is ensured.

Loss in output owing to the hoisting of an empty or partially filled grab has not been shown in practice to be significant. In any case, steps can be taken to ensure that this does not happen by measuring the payload soon after hoisting commences and, if necessary, returning the grab for another bite.

The above developments, together with a compensating device which avoids the grab being plucked from the bottom before closing is complete, enable the dredging equipment to be fully effective in sea-going or swell conditions.

On the application of various types of dredging equipment, it is generally agreed that no one type is universally suitable. The grab dredge is perhaps most suited to operations which are beyond the practical or economic depths for bucket-ladder or suction dredges. There are, of course, many considerations other than dredging depth to be made in the selection of plant, and overlapping of spheres of activity is inevitable. I would suggest, for example, that the grab dredge merits serious consideration alongside the bucket-ladder dredge even at operating depths less than the latter's present practical limit of approximately 150 ft, particularly in view of its smaller capital cost and ability to cater effectively with swell conditions.

The problem of intermittent feed to treatment plant should not create any difficulty provided a hopper with a capacity of one or two grab-fulls is used from which the material is drawn at the correct speed. It is also interesting to note that one sand and gravel company using suction dredging up to a depth of 60 ft decided to employ grab dredges when it was required to extend operations to a depth of 80 ft. The consideration here was one of operating costs—stated to have been approximately halved.

Mr. D. W. Saunders' reference to the Decca 'Hi-Fix' position-indicating equipment prompts me to comment on the justification of the cost of such equipment. I think it should be stated that the equipment provides an accurate permanent record of dredging activities, including depth, with the ability to return precisely to any particular chosen point for a comparatively modest capital cost. The suggested linking of the position co-ordinates to Voith-Schneider propellers via a small computer is, of course, a little more complex, but the advantages of being able to dispense with mooring lines completely will be obvious. Here again, it is thought that in many applications the end will justify the means.

DISCUSSION AT INFORMAL MEETING

A number of Members attending the symposium were anxious to continue the discussion of various alluvial mining problems raised during Sessions 1 and 2. The following is based on a report kindly submitted by Mr. Patrick H. O'Neill, who acted as *Chairman* of the Informal Meeting.

Mr. Patrick H. O'Neill: A group of those particularly interested in alluvial mining assembled in a separate meeting room during the symposium. The meeting began with a discussion of the relative advantages of hydraulic dredges, airlifts, bucket ladder and grab dredges. The use of hydraulic and airlift dredges for diamond mining and their special advantages under certain conditions was commented on by Mr. J. C. C. Hill, who also discussed the possibility of hydraulic and bucket dredges working together where the hydraulic dredge would clean up the uneven bottom. Considerable comment and discussion then arose on the difficulties of cleaning values from very rough bedrock with any type of dredge and Mr. Hill reported on the use of divers for cleaning bottom in diamond mining, although he recognized that such effort might not generally be economical in gold or tin dredging, but certainly could be economic in certain circumstances.

Mr. J. A'C. Bergne commented on the limestone pinnacles encountered in some areas in Malaysia and said he thought it practically impossible to clean between those with either bucket-lines or suction dredges unless it were possible to have a joint in a suction line which would permit vertical lowering and raising of the lower section into the pockets between the pinnacles. Mr. Hill pointed out that a knuckle arrangement had been used in diamond mining in Africa to permit cleaning potholes. In view of the difficulties in cleaning the uneven bottom in tin dredging in Malaysia and the consequent loss of values as evidenced by later native mining in dredged areas, considerable discussion ensued as to the possible use of a suction or airlift unit in conjunction with a bucket dredge, the suggestion being to clean the bottom of each cut after the bucket dredge moved out of a cut. To avoid duplication of recovery plant the suction or airlift unit would pump to the bucket dredge washing plant.

The possibilities of grab dredges for mining operations were then discussed in detail. The lower capital cost in relation to bucket dredges for digging comparable depths was presented as one of the principal advantages. The possible speeds of hoisting, the developments in automation for ensuring proper positioning and full loading and the potential depths for grab dredges were very fully presented by Mr. R. Mohr and Mr. A. L. Stewart.

Mr. G. M. Beaton and Mr. C. J. Simpson advised that a grab dredge with 4-yd^3 (nominal) grabs handling 100 000 to 120 000 yd^3 per month had been in use in West Thailand since 1957 with good results. They reported that it had been developed for use at sea as a self-propelled and self-contained unit. Pictures of the unit showed it to have more the appearance of a ship than a conventional dredge. It dug approximately 20 ft of alluvium lying under 80 to 100 ft of water and the use of the grab dredge proved that it was possible to work a dredge at sea and a conventional bucket dredge of 14-ft^3 capacity had subsequently been working successfully in the same general area.

Mr. Beaton stated that he did not consider the grab dredge a real competitor to a bucket dredge, but did consider it a substitute in certain circumstances. He further suggested the possibility of a combination bucket-line and grab dredge for operating in depths greater than those for which bucket dredges had yet been developed, the idea being that the bucket-line unit would dig down to 150 ft or the maximum depth of the bucket ladder and then the grab units would take over for whatever depth was required. It had previously been stated by the operators of grab units that they could dig to 200 ft or more.

Mr. Beaton replied to questions by stating that no particular problems had been encountered in cleaning up the bottom of the area with the grab dredge and that a system of operation had been worked out that resulted in satisfactory filling of the grabs. He further stated that one of the main problems had been the intermittent feed to the recovery plant, although that had been fairly well solved with a feeder system. They were, however, developing a different system.

A general discussion on the anchoring and positioning of dredges digging at sea then followed. Radio beacon fix systems and echo-sounding devices for face contour information were mentioned, but opinion was that the initial high cost and the high 'operator-alertness' required were the major drawbacks.

Mr. M. A. Brooke then spoke of the high tonnages being handled by dredges, the resulting economics and the need for improvement in recovery systems. Mr. I. R. M. Chaston mentioned the problems in determining actual recoveries and actual losses, especially in relation to the fines in tin dredging, and spoke of test work that had been and was being done to improve recoveries. He also referred to the use of cyclones on dredges to deliver a de-slimed feed to the jigs as a means of improving recovery. Further discussion followed on various splitter arrangements for fore and aft distribution of jig feed and on dewatering systems.

The problems in prospecting and evaluating alluvial ground, especially in the open sea and rivers, were then considered. The use of caissons, airlifts and other methods were discussed.

Also shown at the meeting were pictures of the Ayer Hitam No. 2 dredge, which dug 153 ft below the water-level with 18-ft^3 buckets. Of particular interest on the dredge was the use of cyclones to dewater the trailing for stacking far enough from the stern to avoid re-dredging.

Authors' Replies

(Papers 5–9)

Mr. J. K. Broadhurst and Mr. D. J. Batzer: From Mr. I. R. M. Chaston's comments (pp. 199–201), regarding core measurement in tailings it would appear that some confusion has arisen due to terminology. The dredge recovery of 65–75 per cent referred to the actual tin concentrates won from the re-dredging of old tailings as compared to the estimated tin content of the 'Ore Reserves' in the old tailings as calculated from bores before any tailings factor had been applied to bores in tailings.

A few bores will not give a reliable value of the feed to a dredge any more than, say, stope samples would provide the daily mill feed for a reduction plant. To obtain the true value of the feed to a dredge is not easy and in dredge-sampling programmes it has been our procedure to sample all the tailings products which, together with the tin concentrates won, are taken to represent the value of the feed for the period of the sampling. A screen oversize sample is very difficult to take but, of course, does not affect the value of the feed to the primary jigs.

As the sampling is usually carried out when the dredge is 'treating' as opposed to 'stripping', it will be appreciated that the value of the feed to the jigs is considerably higher than the average bore value, which includes the 'barren overburden'.

It can be said in general that losses of the total free recoverable cassiterite in the alluvium on a dredge are of the following order:

	%
Jig losses	3–7
Screen oversize losses (free ground)	2–3
Spillage from the buckets and clean-up of soft flat bedrock	5
	10–15

Tailings, which it is considered worth while to revalue, would probably have been deposited by less efficient dredges some time ago and the actual total dredge loss might well amount to more than 15 per cent.

Mr. Chaston appears to have misread the example given on page 101 in that he ignored the words 'excavated without treatment'. The question of plant recovery does not enter into the calculation.

We thought that it had been made clear throughout the paper that recoveries and results referred to free recoverable cassiterite, although a brief section of the paper dealt with the total tin content of the alluvium. From tests on bore samples total tin losses amounted to approximately 40 per cent. As there would be a further loss on treating on a dredge, this is in line with the 50 per cent recovery of total tin quoted by Mr. Chaston.

While agreeing with Mr. Chaston that it would be academically sound to give recoveries in terms of total tin, in practice that would be a very laborious process as applied to alluvial mining since practically every result would have to be qualified by a recovery test. It is known in some properties that the +10-mesh BSS fraction of the jig tailings, amounting to some 120 000 yd^3/month, contains locked tin, but one cannot envisage this quantity being reduced by grinding on a dredge to liberate relatively small amounts of physically locked cassiterite.

The 'Barren Overburden' section of Table III (p. 103) states that the free recoverable cassiterite obtained was from the *dulang* tailings, decanted slimes and tin dressers' tailings and this included the free cassiterite recovered from the *amang*. Only *amang* was recovered from the *dulang* and consequently the tin dresser only produced *amang* and a tailing. Because of the absence of a concentrate produced in the field this section of the ground is classified as barren. Individual losses ascertained in the laboratory in the 'barren overburden' were combined for the sake of simplicity; as they are apparently of interest, they are given here (Table A).

TABLE A.—*Free recoverable cassiterite in 'barren overburden'*
(concs. in g 72 per cent Sn)

Bore no.	*AG* 11			*S* 15			*AC* 39		
Mesh BSS	+300	−300	Total	+300	−300	Total	+300	−300	Total
Dulang tailings	0·080	0·008	0·088	0·027	0·163	0·190	tr	tr	tr
Decanted slime	0·069	0·092	0·161	0·024	0·175	0·199	0·009	0·028	0·037
Amang	0·005	0·003	0·008	0·245	0·006	0·251	nil	nil	nil
Tin dressers' tails	0·153	0·019	0·172	0·074	0·123	0·197	nil	nil	nil
Total	0·307	0·122	0·429	0·370	0·467	0·837	0·009	0·028	0·037
Overall value approx. kati/yd^3	0·02			0·01			0·007		

With regard to the assay certificate (Table IV, p. 104), certainly, if the tin dresser is not very skilful and in order to show a high-grade concentrate washes a lot of cassiterite into the *amang*—subsequently recovered, of course, in the assay laboratory—then a high assay adjustment factor will result and assay adjustment factors of 200 per cent have been known. As stated on pages 105–6, the *amang* : cassiterite ratio can affect the assay adjustment factor, but usually a very high assay adjustment factor results when the tin dresser washes to a high-grade concentrate of, say, 75–76 per cent Sn even if his 'loss' in the *amang* is low. It is considered better practice to train the tin dresser to wash to a grade of about 60–65 per cent Sn as losses in tailings are then kept low and an assay adjustment factor nearer to 100 per cent—desirable in monitoring field results in a boring programme—usually results.

Assay certificates for Bores *S* 15 and *AC* 39 of Table III are given (see Table B).

TABLE B.—*Assay certificate*

Bore *AC* 39	Field concentrates	
	Assay, % Sn	Weight, g
(a_1) Field weight of concentrates		33·97
(a) Laboratory weight of field concentrates		33·79
(b) Assay of field concentrates per cent Sn	67·6	
(c) Equivalent weight of 72 per cent Sn conc. $= \frac{a \times b}{72}$	72	31·73
(d) Assay and weight of total *amang*	0·60	857·00
(e) Weight of *amang* taken for magnetic separation		300·00
Weight of magnetics		145·00
Weight of non-magnetics		155·00
(f) Weight of final concentrates from retreatment non-magnetics		2·74
(g) Hence weight of tin concentrate recoverable from total *amang* $= \frac{f \times d}{e}$		7·83
(h) Assay of tin concentrate (f) recovered from *amang*	65·1	
(i) Equivalent total weight 72 per cent Sn conc. recoverable from total *amang* $= \frac{g \times h}{72}$	72	7·08
(j) Total equivalent weight of all conc. @ 72 per cent $= c + i$	72	38·81
(k) Assay adjustment factor $= \frac{j \times 100}{a_1}$	114·25 per cent	
say	114 per cent	

Mr. C. J. A. van Lummel refers (pp. 209–10) to the possibilities of determining tin content by means of the spectrograph. The Geological Survey of Malaysia are currently carrying out research with a Philips

X-ray Spectrograph to determine the accuracy with which this instrument can determine the Sn content of samples. This method, however, like normal chemical assay methods, still gives the total tin content.

At the present moment there is a great need for the development of a rapid method which will report the amount of tin, recoverable by existing mineral dressing methods, in an alluvial sample, as compared with the total tin determined by chemical assay methods.

An assay value of 0·0003 per cent Sn (3 ppm) is equivalent to a value of about 0·01 kati/yd^3 SnO_2 72 per cent Sn, which is of the order required to assess bore-sample loss and jig tailings loss on dredges. (A primary jig treating a grade of ground of 0·20 kati/yd^3 and making, say, a 95 per cent recovery would have a tailing loss of 0·01 kati/yd^3.)

McDonald, Lim and Wong[1] discussed the correlation of chemical tin assays and free recoverable cassiterite and Williams[2] has suggested grain counting to establish the value of a sample in terms of free recoverable cassiterite. Both these methods are dependent on the need to obtain a concentrate in the first place.

A few tests have been carried out by the authors' Company in comparing geochemical assays[3] with samples dressed by hand methods, but no useful correlation has been observed.

REFERENCES

1. McDonald, G. C. R., Lim, L. S., and Wong, Y. L. Relationship between the chemical assay of alluvial tin ores and their recoverable content. *Trans. Instn Min. Metall.*, **72,** 1962–63 (*Bull. Instn Min. Metall.* no. 679, June 1963), 661–8.

2. Williams, F. A. Recovery of fine alluvial cassiterite: Correlation of bore valuations with plant-scale recovery. *Trans. Instn Min. Metall.*, **70,** 1960–61 (*Bull. Instn Min. Metall.* no. 648, Nov. 1960), 49–69.

3. Stanton, R. E., and McDonald, Alison J. Field determination of tin in geochemical soil and stream sediment surveys. *Trans. Instn Min. Metall.*, **71,** 1961–62 (*Bull. Instn Min. Metall.* no. 659, Oct. 1961), 27–9.

Mr. Patrick H. O'Neill: Mr. I. R. M. Chaston comments (p. 201) on flying through the mountains rather than over them and on the additional cost of the operation due to such hazards. It is true that the pilots fly a winding course through ravines, barely clearing the lowest points at 16 500 to 17 000 ft, and often in very bad weather. On my first trips through these passes the pilots were using only their wrist watches and the use of a stop-watch is a later development, Many men have died in crashes often caused by attempting to fly through the narrow passes in bad weather. Although we used, and to some extent still use, commercial service flying from La Paz over the

Cordillera to our operating site, we built an airstrip on the eastern foothills at the end of road transport and had only a 40-mile flight at low altitude to transport the dredge and other camp and construction equipment. Two tri-motored Northrop aircraft capable of payloads up to six tons were used in the airlift and these are still in use supplying the operation. Air supply is expensive and certainly adds greatly to the cost of operation. The road is now being extended to the working area and an operating saving of $80 000 to $100 000 per year is anticipated when the air supply can be terminated.

Mr. Chaston's assumption on using a ten year write-off period for depreciation is correct and this is customary in the operations both in Colombia and Bolivia. As he surmises, the very low depreciation figures for the dredges in Colombia do indicate that the dredges are depreciated up to the legal limit (10 per cent) and the low figures used now for depreciation are for modifications or major equipment replacements. Mr. Chaston suggests that in areas of disturbed political conditions perhaps a shorter write-off period should be budgeted for. I could not in good conscience recommend a substantial investment in any new project in a disturbed area unless it were possible to estimate a return of capital within five years or less. However, I would not recommend going into any new area without confidence that it would be possible to operate for many years. Our associate companies have had their 'ups and downs' during many years of operation in South America, but generally have been treated fairly well. Investment guaranty insurance against expropriation is also carried on the Bolivian property on the off chance of another expropriation similar to that in 1952, but this does not seem very likely. Consequently, a depreciation of ten years is used as it is well within the life of the developed reserves and it also permits a reasonable earnings figure.

Mr. C. J. Simpson enquires (p. 208) about running time and percentage bucket capacity allowed when calculating the actual yardage. Running time generally averages 80 to 85 per cent on a yearly basis. All down time is deducted, except major shutdowns or long moves to new locations.

The yardage reported as actual yardage is a measured yardage through survey of the actual dredge cut outline and using average depth dug by the dredge during the period in the various cuts. In other words, the yardage shown is actual bank yardage or ground in place and the natural swell of the ground is not considered. Percentage bucket capacity does not enter into the actual yardage calculations. The percentage of maximum bucket capacity varies considerably with digging conditions. It is always low when digging in rivers with no advantage of caving banks or in tight ground with many boulders. In easy digging some of the dredges average from 80 to 95 per cent full buckets. Last year one averaged 98 per cent and one dredge averaged over 100 per cent for a time when it was carrying a bank of silt that caved and disappeared in the pond. Dredges in rivers or tight digging seldom average over 60 per cent full buckets and occasionally are as low as the 30 per cent indicated for the dredge in Bolivia—which is by far the most difficult dredging I have seen anywhere.

Mr. M. J. Cruickshank: It was indeed very gratifying to note the interest shown in the subject of my paper and I was sorry not to be able to join in the interesting and informative discussion.

Referring to Mr. J. S. Sheppard's comments (pp. 202–3), I agree that the omission from Table II of sand and gravel deposits in the United Kingdom was a major one. This was, in fact, noted prior to publication by the Editor, who included the data on page 146 to which Mr. Sheppard refers. The projected annual production of ten million tons in 1968 given by Mr. Sheppard is certainly substantial, although I was surprised to note the ex-wharf value of approximately $3.00 per ton. The value per ton of sand and gravels in the United States from all sources has been held at under $1.00 since the 1930s, despite rising labour and equipment costs.

Mr. I. H. McLean (p. 203) asked some pertinent questions on drop coring, dynamic positioning and Shell's Nome operation. In general, the drop corer is a very inefficient device for the sampling of sands and gravels from the sea bed. The main drawbacks result from lack of penetration due to compacted fine materials or to obstruction by coarse materials. Retention of the core is a problem with anything but fine mud or clayey materials owing to washing out of the core on the haul to surface. Core retainers which close the bottom of the core barrel after extraction of the corer alleviate this problem to some extent, but they can, and often do, jam open. A success factor of 10 per cent in the retrieval of cores is not unusual and a penetration of one or two feet at most is the order in compacted sands or gravel. Dr. Francis Shepard of Scripps Institute reports much better success in sands with the box corer, developed by Dr. Arnold Bouma in the Netherlands. This consists of a steel box of dimensions about 12 in. × 15 in. × 18 in., mounted on a tripod and made to penetrate the bottom by the release of a heavy weight mounted on the box core shaft. On tensioning the suspension cable a sliding cover is caused to arc under the box, sealing it tightly before it is withdrawn from the sediment. The samples are in most cases undisturbed and coarse materials such as cobbles and sea-floor nodules can be accommodated.

With regard to dynamic positioning, while it is highly successful in maintaining the position of a drilling ship in deep water, even in fairly rough seas the flexibility of the drill string permits a much greater lateral movement than would be allowable in water of moderate depth. By simple geometry the lateral movement permitted in the Mohole operation would be reduced to only 10 ft in water 600 ft in depth. It is doubtful whether this sort of stability could be maintained solely by dynamic positioning at the present time, but in combination with one or two anchors it would probably be possible to come close to it.

Mr. P. W. Dobson's remarks (p. 212) on the suggested linkage of position coordinates from a Decca 'Hi-Fix' position indicator to Voith-Schneider propellers via a small computer are encouraging. He intimates that completely automated positioning may be quite feasible, although sea conditions would no doubt be a dominant factor in practice. The Rockeater currently exploring for diamonds off South West Africa employs two bow anchors and two stern anchors, in combination with a bow thruster giving lateral control at the bow of the ship, and a Harbormaster outboard giving variable control at the stern. Because of the shallow water, less than 200 ft, and the rigid drill stem, a very high degree of stability is required during drilling. This method reportedly fulfils this requirement, at least in wind speeds below 35 knots. The real answer to Mr. McLean's query obviously lies in the interpretation of the term 'fairly troubled waters'.

The operation at Nome, Alaska, is still in the confidential stage and detailed cost figures are not available. During the 1964 drilling season Shell used a new type of rig for drilling from the ice up to one mile from shore. Developed by a Canadian firm, the drill consists essentially of a double casing, the inner one 3 in. in diameter and the outer one about (?) 6 in. which is pile-driven into the unconsolidated sediments. High-pressure air and water jets are used both to loosen the sediment at the cutting shoe and to effect removal of the solid material in the centre casing by airlift. Separation of the air–water–solid mixture from the airlift is controlled by a cyclone. Penetration rates are reportedly very high and recovery of values good.

Mr. J. A'C. Bergne's comments (pp. 204–5) on trawling for minerals are well borne out by published figures. He would no doubt be interested in Mero's recently published book[1] which deals very thoroughly with present knowledge on the distribution and mining of manganese nodules. The method referred to by Mr. Bergne of position-fixing with a sextant is mentioned on page 129, although not in detail. It is certainly a very useful and widely applied method, particularly in small craft working, and is often used by geologists on larger oceanographic vessels to confirm or substantiate the fixes taken on the bridge.

In reply to Mr. M. A. Brooke (pp. 205–6), the exact nature of the difficulties encountered by the Collins operation in cleaning to bedrock are not known, but one can assume that they would be due to unevenness or fissuring of the rock bottom. The raised beach terraces which are mined in the Chameis Bay area generally contain the best diamonds at the base of the undercut sea cliffs and in potholes. After mining, cracks in the schistose bedrock are scrupulously cleaned by hand brushing. It is likely that the submerged deposits are similar and while it is unlikely that sweeping of the sea-floor is carried out, certainly inspection by divers or underwater television, with which Rockeater is equipped, would be quite feasible at the depth in question. Mr. Brooke's further comments on the value of tin operations in Thailand are appreciated. The estimates were certainly very low and should have been nearer the figures he mentions.

The prospecting method for river and offshore alluvials mentioned by Mr. J. C. C. Hill (pp. 208–9) is similar to that employed on the Rockeater prospecting vessel designed and operated by Ocean Science and Engineering off the South West Africa coast under contract to De Beers Consolidated Mines, Ltd. There a 36-in diameter casing is sunk utilizing, alternatively or in combination, airlift, suction lift and venturi for the removal of material. Although there is no report of divers being used consistently in this operation, they are used in guiding the suction head in some of the dredges mining offshore ironsands in Japan.

The written contribution of Mr. G. M. Beaton and Mr. C. J. Simpson (pp. 210–1) is elucidating and very welcome, although it refers to a number of points which call for reply. When the data for the original work on which my paper was based were being collected, Tronoh Mines, Ltd., was one of the Companies which I contacted by personal letter. Their reply was very brief and uncommunicative and referred me to the published literature. Further correspondence on the point failed to elicit any reply and the data presented were therefore gathered piecemeal from various sources. I regret any factual errors presented and thank the contributors for pointing them out.

Referring to the Aokam Tin, Ltd., grab dredge, Romanowitz[2] gives an operating depth of 215 ft. He also gives depths of 155 ft, 135 ft and 116 to 130 ft for various ladder dredges. While it is true that a dredge to work to a depth of 250 ft below water-level would require enormous capital expenditure and very high recoveries to justify its construction, it would nonetheless be feasible. The upper limit of capacity for a bucket-line dredge of 3000 yd^3 per hour given in Fig. 2 (p. 131) is, in fact, a low estimate using Romanowitz's proposed 54-ft^3 bucket with a line speed of 35 buckets per min. Again referring to Fig. 2, the upper limit figure of 15 000 yards per hour is indeed an error and should be 1500 yards per hour as noted in the text on page 134. Finally, the information on Aokam Tin's proposed bucket dredge emanates from the *Mining Journal*[3] which states: ‘Aokam Tin, in which Ayer Hitam has a substantial interest, at present operates a grab dredge off the west coast of Thailand; consideration is now being given to the installation of a sea-going bucket dredge. This would considerably improve the productive and earning capacity of the company’.

REFERENCES

1. MERO, J. L. *Mineral resources of the sea.* (Amsterdam: Elsevier, 1964), 312 p.

2. See reference 10 on page 148.

3. *Min. J.*, **257,** no. 6592, Dec. 22 1961, 661.

GROUP II—QUARRYING AND OPENCAST MINING OF MASSIVE DEPOSITS

Session 3—Quarrying Practice (Papers 10–12)

Chairman: Mr. F. Frith *Rapporteur:* Mr. B. G. Fish

622.235:622.271.3

Paper 10

Developments in Blasting Techniques in Opencast Mining and Quarrying

B. J. KOCHANOWSKY, Dipl. Ing., Dr. Ing., M.A.I.M.E.
Professor of Mining Engineering, The Pennsylvania State University, U.S.A.

SYNOPSIS

Recent results of research into and development of drilling and blasting techniques used in surface mining are given. Theoretical and practical aspects of angle drilling are emphasized, various new drills being described. Attention is also drawn to AN–FO and slurry explosives.

In the United States of America almost all metallic ores and rock are produced in open-pit operations and most of this material must be blasted. Blasting techniques determine the fragmentation degree, which greatly influences the productivity and maintenance costs of shovel, truck and crusher. For example, a thirty per cent powder increase enabled a taconite mine to double the shovel output and to increase the lives of shovel bucket and teeth four- and six-fold respectively. Quarry plants often have bottlenecks in the crushing and grinding operations, or are designed without a buffer in the haulage between primary crusher and plant. Improved fragmentation can increase the capacity of the plant without costly modification.

Blasting techniques are determined by two factors—safety and economics. Since bore-hole layout design and the calculation of explosive charges are based on theoretical assumptions, knowledge of blasting theory is particularly important. To improve blasting techniques it is necessary to know how the explosive force acts, how rock resists this force, and how the rock-failure mechanism occurs and can be explained. Because of the tremendous power developed within one millisecond, and the great number of factors directly affecting the technical and economic results, an analysis of the fundamentals of blasting theory is very difficult.

Of considerable interest to industry are the most economical bore-hole pattern, bench height, bore-hole diameter, burden and spacing. The main factors that determine these parameters are cost of drilling and cost and handling of explosive. The final decision will, of course, depend upon the determination of overall minimum costs and the summation of costs for drilling and blasting with those for loading, haulage and crushing. The many factors that determine the economics of blasting create difficulties when a quantitative correlation between cause and effect in blasting phenomena is attempted. Confusion and difference of opinion are therefore common among scientists and engineers concerned with explosives.

BORE-HOLE PATTERN

Recent developments in blasting techniques clearly indicate that the angle drill-hole pattern, in the main, will replace vertical drilling.

Before 1959 there were only three operations in the U.S.A. using angle holes not exceeding 10° or 15° from the vertical. In 1962, however, over fifty companies were either using this method, testing it or planning to introduce it. The trend towards angle drilling is apparent today in Canada, Europe, South America and South Africa.

The following advantages are claimed by companies employing angle-drill methods:[1]

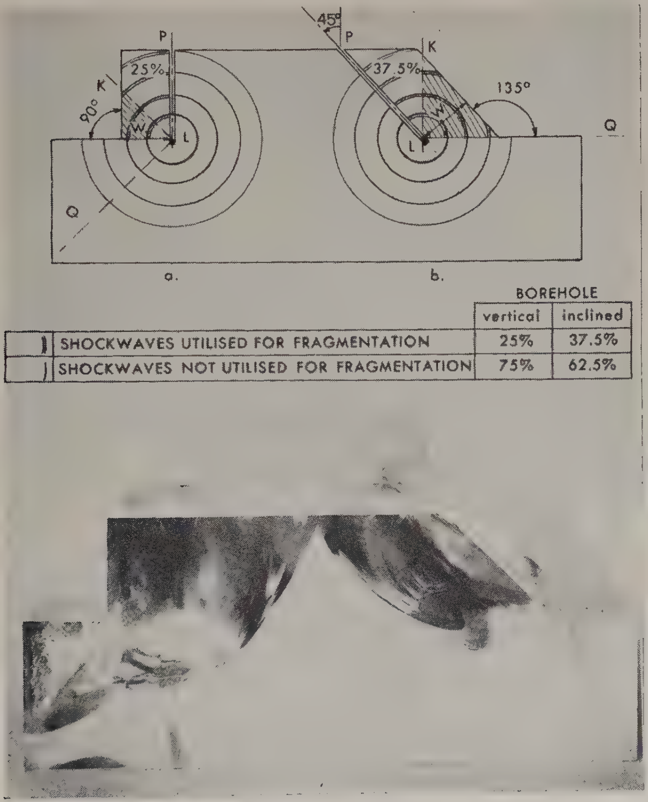
(1) operation for men and equipment is safer;
(2) fragmentation has been improved considerably and cost for secondary blasting greatly reduced;
(3) toe problem, one of the biggest operational problems, is negligible;
(4) difficulties caused by backbreak are practically eliminated: the face is clean with very little overhang;
(5) explosive consumption has been reduced by one per cent per degree of inclination from the vertical;
(6) less drilling per ton rock, due to larger spacing and/or larger burden and less sub-drilling;
(7) less vibration; and
(8) overall costs, including drilling, blasting, loading and crushing, dropped in certain cases by up to 10 per cent.

The advantages of angle drilling have also been verified by Cheshire.[2] It must also be pointed out that Hettwer,[3] who was the first to introduce 30° angle holes in Germany, blasts over 20 ft of burden in a limestone quarry in Honnetal with holes only $3\frac{1}{2}$ in. in diameter. Although these holes are blasted simultaneously with the toe holes, this is a remarkable achievement and can be obtained only with inclined holes.

In Europe, where the bore-hole diameter and burden are small and benches high, the angle-drill method is of particular importance. Because of breakage and fall of rock from the upper edge of the bench face and widespread fracturing of rock, it is difficult or even impossible with vertical drilling to place the drill machine sufficiently close to the bench face line to obtain the desired burden, which is often too big for the bore-hole diameter. Poor fragmentation, toe problem and backbreak therefore result. Angle drilling eliminates all these problems, is safer, and reduces costs.

It has been said that knowledge of blasting theory is of great practical importance and the sudden success of angle drilling is confirmation of that statement. Angle drilling is an old method, but despite the fact that any machine drilling holes up to 4 in. in diameter can be tilted easily the method was practically ignored in the past. Following the explanation of and publicity given to the theory of angle drilling and its advantages, several new drills designed for angle drilling appeared on the American market and the method was introduced by many companies. The theory can be reduced to three main statements concerning the utilization of the explosive energy, the rock resistance and the outbreak angle.

(1) It is generally assumed that the rock is broken by a shock wave reflected from a free face. From Fig. 1 it can be seen that only 25 per cent of the total shock-wave energy is reflected from the free



		BOREHOLE	
		vertical	inclined
▨	SHOCKWAVES UTILISED FOR FRAGMENTATION	25%	37.5%
☐	SHOCKWAVES NOT UTILISED FOR FRAGMENTATION	75%	62.5%

Fig. 1.—Shock-wave propagation through rock generated by detonation of explosive charge at point *L*.

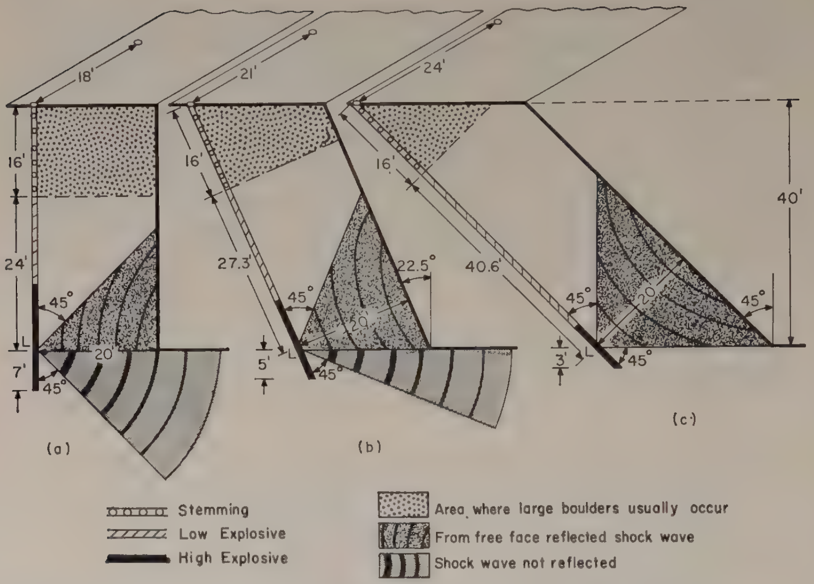


Fig. 2.—Angle holes allow better utilization of explosive energy and thus permit larger spacing, better fragmentation, smaller sub-drilling and require less of expensive explosive at the bottom of the hole.

face when vertical holes are used. By tilting the hole it is possible to guide a greater amount of shock-wave energy against the bench face. From Figs. 1 and 2 it is apparent that in a 45° inclined hole twice the shock-wave energy generated in point *L* is reflected from the bench face than in the vertical bore-hole.

(2) The greater the inclination of the bench, the smaller is the rock resistance against blasting.

(3) The greater the inclination of the bench, the greater is the outbreak angle.

DRILLING MACHINES

Most new drill designs have been developed for angle drilling and larger bore-holes—up to 15 in. in diameter—can now be drilled. A further saving in labour costs is possible because several drills can be placed upon a carrier, but operated by one man. For small units four machines are used and for large units, drilling holes up to 9 in. in diameter, two machines. These machines are supplied with hydraulic positioner* for spacing.

---

* A device having a hydraulic mechanism permitting the change of spacing (distance) between the drilling machines.

Drilling research aims at (1) automation of the drill to permit instantaneous matching of down-pressures and rotary speed with rock type; (2) development of harder material for the bits; and (3) development of drills working with 200 to 250 lb/in^2 air pressure. Drilling penetration has been doubled and bit carbide wear per foot cut by half by increasing the air pressure from 100 to 200 lb/in^2.

It is known that the bottom charge in the bore-hole needs 2·5 to 4 times more explosive than the column charge. The idea of inventing a tool that would permit enlargement of the bore-hole end by mechanical means came to nothing and the only machine that can do this easily is the jet piercer applied to very hard rock, such as taconite. A small jet piercer was recently developed for the purpose of chambering the hole after it has been drilled by conventional drilling machines and uses air instead of oxygen. Research is being conducted to establish in which rock this new procedure might find wide application.

BENCH HEIGHT, BORE-HOLE DIAMETER, BURDEN AND SPACING

It is known that American open-pit and quarry operators prefer low benches, whereas European operators prefer high ones (see Table I) and a theoretical explanation for these preferences can be found fairly readily.

Owing to the greater rock resistance at the bottom of the bench the bottom charge requires 2·5 to 4 times more explosive per ton rock than the column charge. Therefore, assuming the same burden, the higher the bench the less explosive is consumed per ton rock. The smaller bore-hole diameter used in Europe leads to the burden selected being smaller than that used in American quarries. Small burden, B, and a high bench, H, creates a high ratio between bench height and burden, H/B, another factor responsible for a smaller explosive consumption in European quarries. It can be seen that in Europe the bench height, burden, and bore-hole diameter are selected to minimize the explosive consumption, because of the high price of explosive in relation to labour costs. Since the latter increase from year to year, however, whereas the price of explosive remains comparatively constant, this relationship also changes, and European operators have already begun to select lower bench faces. Their main reasons for doing so are the greater safety ensuing and smaller rock-pile height, which permits more effective use of a drop ball and a higher performance of mechanical shovels, as well as of other mechanical equipment. A low rock-pile height can, of course, also be obtained on high benches, but a higher explosive consumption is required to obtain a greater throw and this might be costly when explosives are expensive.

A certain relationship exists between the bench height and bore-hole diameter. Usually—but not necessarily—the higher the bench, the greater is the burden selected. A closer relationship exists between bore-hole diameter and burden. With a small bore-hole diameter a large burden can also be blasted, but the explosive consumption per ton rock will increase with increased burden and/or rock fragmentation will become poorer. Normally, smaller spacing and the help of the toe bore-holes are required to pull out a large burden with small bore-hole diameter.

The price of a drilling machine, as well as the bit, increases in linear proportion to the bore-hole diameter. The labour cost, when drilling is paid by the hour, is a fixed cost. Owing to a heavier design of drill for larger bore-hole diameter the rate of penetration for larger bore-holes is approximately the same or even greater. The labour cost per foot of hole for large and small bore-hole diameters is therefore almost the same. The bit life expressed in feet of hole drilled, until the bit is worn out, is greater for a larger bore-hole diameter (Fig. 3). The total drilling costs per foot of hole are easy to calculate: a bore-hole drilled with a rotary machine of twice as large a cross-sectional area costs not 100 per cent more per foot drilled but, depending upon bore-hole diameter range, only up to 30 per cent more.

Burden and spacing are generally selected so that a double cross-sectional area of a bore-hole blasts twice as much rock volume per foot

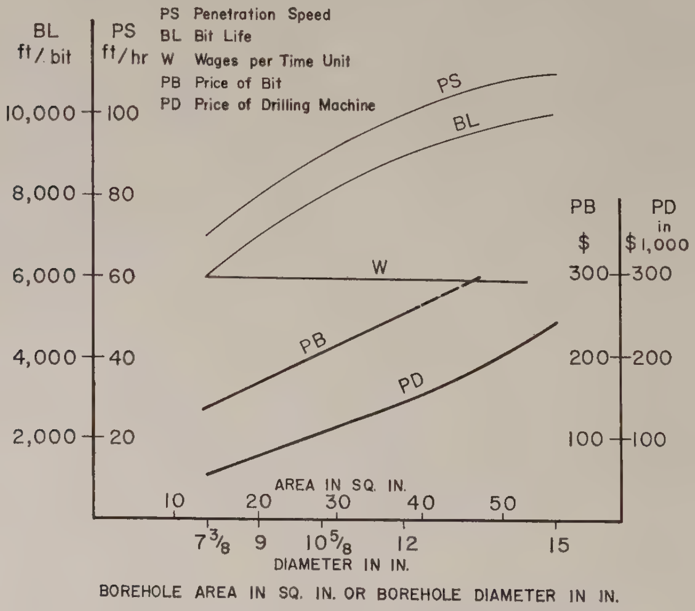


Fig. 3.—Parameters of rotary drilling in coal stripping versus bore-hole diameter.

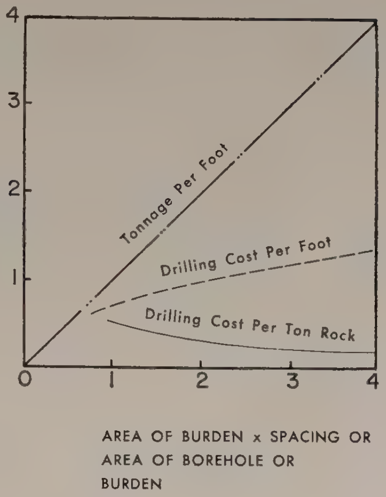


Fig. 4.—Drilling costs and tonnage versus area of bore-hole.

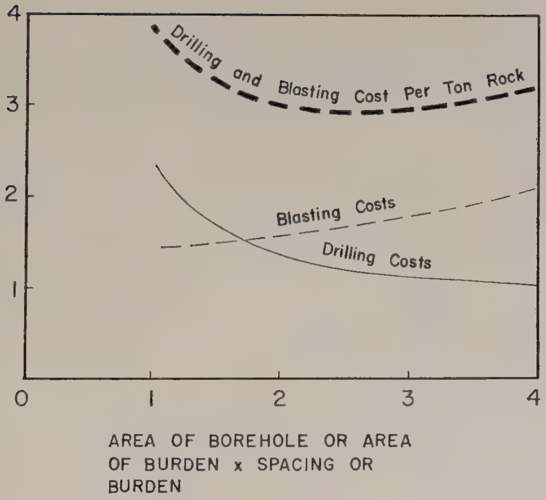


Fig. 5.—Drilling and blasting costs versus area of bore-hole.

of hole. From the point of view of drilling costs per ton rock it would therefore be cheaper to use a larger bore-hole diameter (Figs. 4 and 5). In case the powder factor is kept constant, however, the larger burden and spacing would create fragmentation problems since larger rock pieces would be obtained, which can, up to a certain point, be handled satisfactorily by larger loading, hauling and crushing equipment. Another way of influencing the fragmentation is the powder factor. How much extra explosive is necessary for a larger burden and spacing in different bench heights to produce the same fragmentation is difficult to determine and exact figures are not yet available. Without this information the problem cannot be solved (Fig. 5) and its complexity means that the bore-hole diameter used in industry is not always the most economical one.

In general terms it can be said that, so long as the fragmentation does not create a problem for the operator, the larger the bore-hole the more economical drilling and blasting become. As mentioned, several machines which drill 15-in diameter holes have been introduced since 1962. However, where, due to structure and mechanical properties of rock and to the size of equipment used fragmentation problems exist, a smaller bore-hole diameter might be a better solution. Nevertheless, it must be remembered that the smaller the bore-hole diameter, the less explosive can be loaded per foot of bore-hole. Also the detonation speed of explosive is slower and the air space in the bore-hole is relatively larger if explosives in cartridge form are used (Fig. 6). Therefore with a smaller bore-hole the loading density, the explosive gas pressure per area of bore-hole and the shock transmitted to the rock are all poorer (Figs. 6 and 7). Considerable differences of opinion therefore exist concerning the most economical bore-hole diameter. In the U.S.A. most companies assume that the smaller bore-hole enables better results to be obtained. Some large mining companies, however, working in similar conditions have adopted a larger bore-hole diameter.

As already mentioned, when the area of bore-hole is doubled, the rock volume blasted per foot of hole is generally also doubled by adjusting the product of burden times spacing. Since the drilling cost per foot of hole increases degressively with increased bore-hole diameter, the drilling costs per ton of rock would decrease rapidly with larger bore-hole diameter. If instead of doubling the volume with a bore-hole diameter twice as large a smaller volume of rock per foot of hole is blasted, the drilling cost per ton of rock will increase, but the blasting efficiency and rock fragmentation will be improved. Thus by optimizing the rock volume per foot of hole with an increased bore-hole diameter the overall cost might be lessened.

It is customary to select spacing equal to or smaller than the burden. The theory of blasting and tests on both a small and large scale[4] shows that it is more advantageous to have a spacing larger than the burden. The ratio of spacing to burden may reach 1·8, under favourable conditions, but is usually in the range 1·2 to 1·5. An increasing number of companies now use a ratio greater than unity of spacing to burden. When the direction of main fractures is more or less perpendicular to the bench face and these fractures are filled with sand or clay, a great ratio between spacing and burden may not be desirable since a poor fragmentation might be produced.

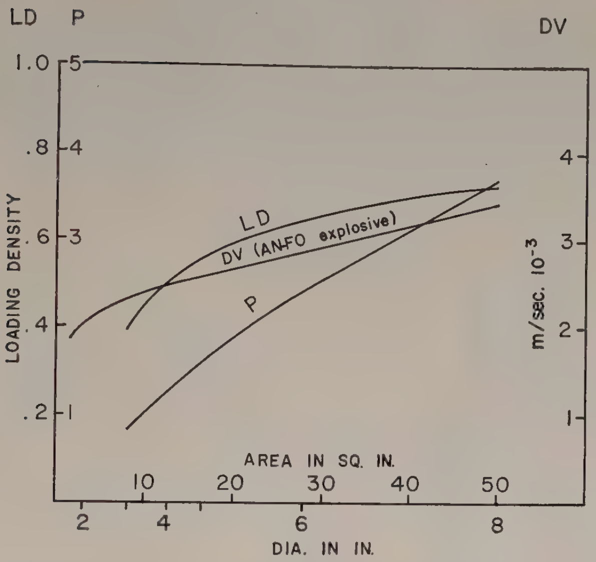


Fig. 6.—Loading density (*LD*), detonation velocity (*DV*), and relative weight of explosive per area unit of bore-hole wall (*P*) versus bore-hole diameter.

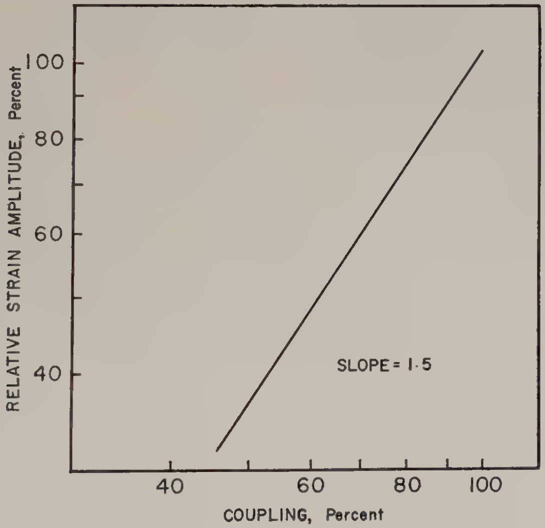


Fig. 7.—Strain transmitted to the rock by the explosive charge is approximately proportional to the 1·5 power of the 'coupling', i.e. the ratio between the charge diameter and the bore-hole diameter.

Benefits of AN–FO and Slurry Explosives

Table I shows the difference of explosive consumption and number of workers required in modern European and American quarries. The explosive consumption per ton of rock in Europe is much lower than in the U.S.A.; however, the number of workers is much higher. This is true even when geological conditions, as well as the size of shovel and crusher, are the same. The ratio of the cost of explosive to that of wages is one reason for this state of affairs. According to the data given in Table I the value of a man-shift, expressed in kg of explosive, varies between 50 and 700 in the U.S.A. and between 12·3 and 48 kg in Europe. Assuming the average price for explosives the difference in

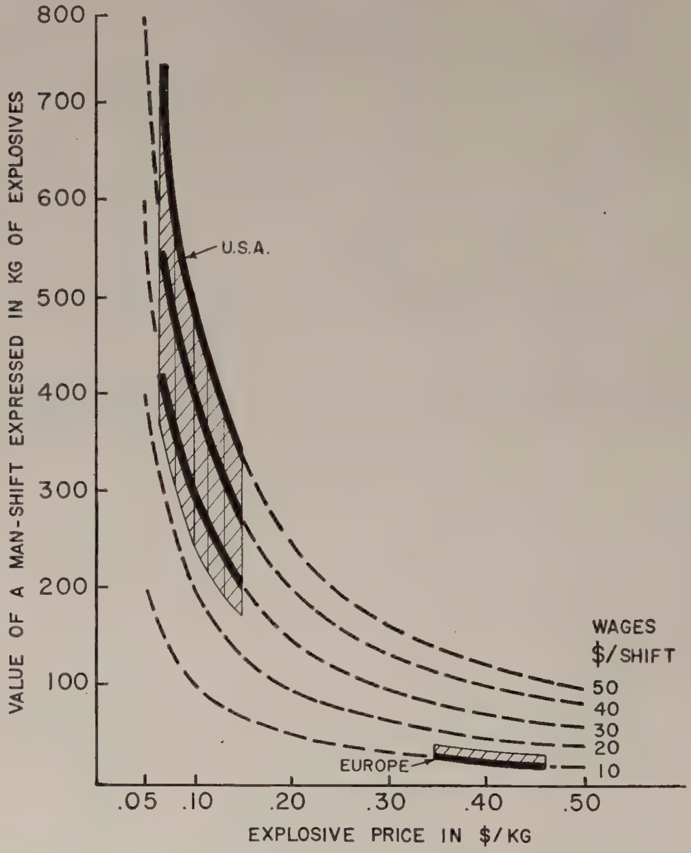


Fig. 8.—Value of a man-shift expressed in kg of explosive.

'value of a man-shift expressed in kg of explosives' in the U.S.A. and in Europe will be even greater, as is shown in Fig. 8. Cheaper explosive is the first answer to this question.

Ammonium nitrate and fuel oil mixture (AN–FO) and slurry are explosives that could reduce the cost of explosives in Europe. It is even more favourable when these explosives are manufactured by the mining companies themselves, which consists in mixing of different components. The manufacturing process is very simple and safe. As Fig. 8 shows, when cheap explosives are available the company is able to use more explosive per ton of rock and, with the better fragmentation obtained, to reduce costs in secondary blasting, loading, haulage and crushing.

TABLE I.—*Characteristic data for operations in European and American rock and mining industries**

	Europe	*U.S.A.*
Bench height, ft	50–100	30–50
Bore-hole diameter, in.	3–4 (6)	(4) 6–15
Cost per man-hour, $	1–1·50	2·50–7
Price of explosive, $/kg	0·25–0·65	0·08–0·40
Man-shift per 1000 t/day	2–3	0·4–0·6
Explosives consumption, g/t	50–100	100–200 (400)

* Bracketed numbers represent occasional limits.

AN–FO explosive has already been introduced in several European countries and is publicized in the European literature.[5] The shape and size of ammonium nitrate, whether flaked or prilled, determine the explosive strength. The explosive is not particularly strong, but because it is used in loose form it fills the bore-hole completely and therefore has a high loading density and a good blasting effect. It is used normally for column charges. Scott claims that an AN–SN–FO mixture (40 per cent sodium nitrate) gives approximately 60 per cent more power than a straight AN–FO mix alone. Since bulk sodium nitrate is as cheap as bulk ammonium nitrate this discovery may be of benefit to the industry in the future.

Compared with AN–FO explosives, slurry explosives have a 50 per cent higher detonation velocity and double the specific gravity, and are not sensitive to water. They are very powerful and, therefore, used for bottom charges and/or in hard rock. The addition of metallic powder enables the explosive to reach very high strength. In practice it is used in cartridge form, but because of its gelatinous consistency it can be pumped into the bore-hole, a method that will probably be used widely in the future. There are several compositions of slurries, one being a mixture of AN, TNT and water with a dash of guar gum. They can be brought ready for use or mixed by the mining companies themselves.

Because of the higher power and greater density of slurry explosives, smaller bore-hole diameter and/or greater burden and spacing can be applied. This is of great importance in hard rocks where drilling costs are very high. The Iron Ore Company of Canada found that spacing could be increased from 8-ft by 8-ft patterns to 12-ft by 12-ft patterns with a 55 per cent saving in drill costs. Compared to AN–FO the explosive costs increased 68 per cent, but overall costs decreased 42 per cent.

Results obtained through research and blast tests by The Dow Chemical Co., E.I. Du Pont de Nemours and Co. and others indicate that in future metallic slurries will permit a much higher tonnage per foot of hole and the desired fragmentation to be obtained.

AN–FO explosive was introduced in the U.S.A. nine years ago and slurry explosive about five years ago. AN–FO prices average between 4 and 5 cents per pound and slurry between 7 and 18 cents. Other commercial explosives of the same strength are two to three times more expensive. These new explosives have helped the American mining and rock industries to achieve a considerable cut in costs and the same saving could happen in Europe. The cheaper the explosive, the larger the amount of explosive that can be used per ton of rock. Table I shows that in the U.S.A. twice the amount of explosive is utilized per ton of rock compared with European consumption. As the powder consumption increases, manufacturers will be able to sell a greater amount of explosives. By selling explosive more cheaply manufacturers would encourage the mining and rock industries to spend more—to the benefit of all concerned. The alternative is for companies engaged in the mining and rock industries to manufacture explosives themselves. In some countries monopolies and/or other legal obstacles make such manufacture difficult or impossible, but it is doubtful that such artificial barriers can hold for long.

CONCLUSION

Geological and economic conditions dictate the technique to be applied in each country. It is evident that angle drilling and cheap explosives such as AN–FO and slurry are of great importance to the rock and mining industries. These are safer than other explosives available commercially and manufacturing procedures are safe and easily acquired. European mining and rock industries could lower their blasting, loading, hauling and crushing costs considerably by their use and by the wider adoption of angle drilling.

REFERENCES

1. Progress made with a research project. . . Angle drilling for surface mining under the direction of B. J. Kochanowsky. *Miner. Ind., St Coll. Pa*, **32**, no. 8, May 1963, 1–4.

2. CHESHIRE, M. A. The advantages of inclined holes at hard rock faces. *Quarry Mgrs' J.*, **47,** May 1963, 177–86.

3. HETTWER, A. Betriebskonzentration als Aufgabe bei der Gewinnung in Kalk- und Dolomitsteinbrüchen. *Nobel Hefte Schiessw.*, **28,** no. 3, May 1962, 93–107.

4. BAUER, A. Application of the Livingston theory. *Colo. Sch. Mines Q.*, **56,** Jan. 1961, 172–82.

5. MACKINTOSH, I. B. Progress in ammonium nitrate blasting. *Mine & Quarry Engng*, **29,** no. 11, Nov. 1963, 485–9.

Calculation and Application of Heading and Deep-hole Blasts for the Development of New Quarry Faces

J. LUKEŠ, Doctor of Law, Dipl.-Ing.

Mine and Quarry Inspector, Central Bureau of Mines; formerly of the Association of National Enterprises of Stone and Gravel Industry, Prague, Czechoslovakia

---

SYNOPSIS

The paper discusses the methods of blasting most suitable for the large-scale production of stone in Czechoslovakia to meet the increased demand. Quarry faces must be prepared for the method chosen and examples are given of planning heights and siting charges of benches for primary blasting in certain combinations to produce suitable conditions and keep to the minimum the haulage distances involved in the transport of broken rock.

Methods of calculating the charges necessary, according to the consumption of explosive or effects of detonation on surrounding rock, are explored. Formulae in current use in Czechoslovakia for determining the charges for mass blasts, and their accuracy and limitations, are examined. Methods of computing the charges required for corrective blasting are also discussed.

In the past fifteen years the demand for natural stone in Czechoslovakia has risen considerably, requiring the production of sufficient quantities for building construction, roads and railway ballast. Large producers operated where limestone was quarried, other classes of stone being won predominantly in small quarries, where selective mining was practised to give a wide assortment of products. Generally, quarrying was by blasting individual shotholes, which gave rise to cramped workings, often poorly developed.

The increased demand for stone could be met only by a radical transformation of quarries into specialized plants for bulk production. Modernization has also enforced changes in the methods of quarrying, particularly in the introduction of large blasts to secure a sufficient stock of broken rock. Such mass production was founded on chamber and deep-hole blasts.

The situation was typified by the following circumstances:

(1) closed, horseshoe-shaped quarry with short producing face;
(2) unsuitable slope of quarry face;
(3) quarry faces from 40 to 120 m high;
(4) floors unsuitable for the use of portable drilling equipment; and
(5) lack of suitable and efficient drill rigs.

Under these conditions chamber or heading blasts became the method of blasting most extensively used. To create conditions suitable for the application of deep-hole blasts required a considerable time for the preparation of quarry faces, but such blasts are now gradually being introduced in all quarries where heading blasts were first applied and for preparing faces for high-capacity production.

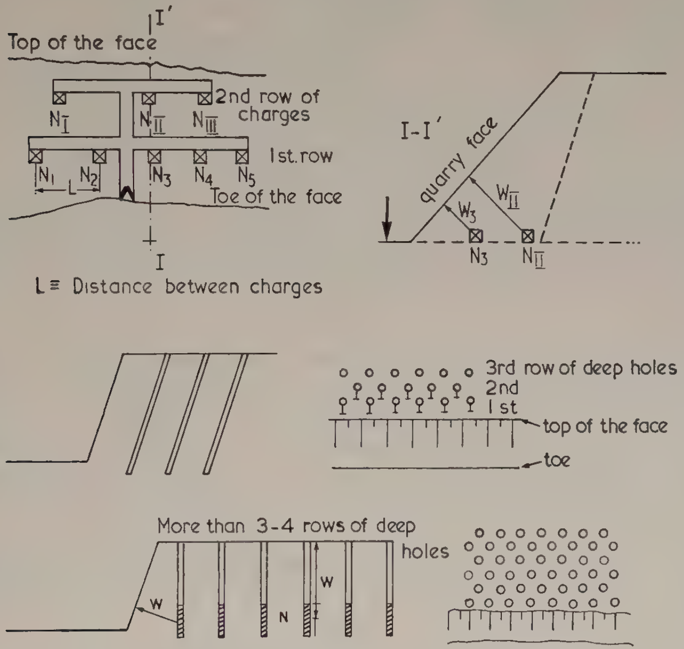


Fig. 1.—Blasting methods—heading, deep-hole and square blasts.

N = charges in chambers
(*a*) Chamber blast or heading blast
(*b*) Curtain blast or deep-hole blast
(*c*) Square blast

PREPARATION OF QUARRY FACE

To change small quarries into medium- or high-capacity units it is first necessary to create a number of independent workplaces, either by extending quarry faces sideways or, with a high but narrow face, by benching while extending the face. The first alternative requires careful calculation of the chamber blast charges, while the second case requires correct evaluation of all the factors from both the technical and the economic point of view.

Various methods were tried out in Czechoslovakia. Some enterprises tried blasting the face to its full height to save the cost of roads to separate benches. Blasting was done using two chamber blasts, one over the other, detonated simultaneously or separately, a method which endangered men and machines by falling stone. Later a combination of a deep-hole blast for the upper part of the face with a chamber blast for the lower bench was tried, some quarries attempting to create benches by this method of blasting.

Generally speaking, the heights first chosen for the upper bench were too small, say 15–20 m. As rock in the uppermost strata is often weathered, the proportion of low-quality stone in the total production was then too high. It is the lower portion of the face which usually yields stone of the best quality and this was still broken by chamber blasts, which necessitated a considerable amount of secondary blasting and caused heavy wear and tear of mechanical equipment.

Experience gained from this early work has led to an entirely opposite approach. Now it appears better to use a combination in which the top bench is of the greatest thickness and broken by chamber blasts, while the lower bench or benches are thinner and are taken out by curtain or square blasts.

For example, a quarry face of 80 m total height is divided into three benches. The top bench, 40 m high, is won by chamber blasts, while the lower part is divided either into two benches 20 m thick, mined by curtain blasts, or into one top face 25 m high, broken similarly, and a still lower portion 15 m in thickness broken by square blasts. In Czechoslovakia the ratio of costs for heading and curtain blasts is 1:1·5 up to 1:2. With the drilling rigs currently used the costs rise approximately by 2 per cent for each linear metre of depth of bore-holes.

The combination of primary blasting methods just described possesses the following advantages.

(1) Breaking the top bench by heading blasts does not require what might be a costly levelling of the ground above the deposit to make it suitable for the movement of drilling rigs. Removal of the overburden is all that is required.

(2) The height of the top face is determined by the technical parameters of heading blasts. It is only necessary to adjust the ratio of the height of the face to the most suitable range of shattering effect of the main charges; this is given primarily by the petrographic properties of the rock and its attitude. In this connection the layer of weathered rock in the top slice must not be overlooked as material under the overburden is generally weaker and splinters more readily. The height of the face to be blasted by heading blasts can be increased by the thickness of the weathered layer of rock with no danger of producing an increased percentage of oversize. There is also less likelihood of dangerous over-hang occurring.

(3) With a relatively high face it is possible to keep down the percentage of stone of lower quality within the limits stipulated in the respective standards.

(4) The choice of the maximum height of the top bench reduces considerably the distance through which the stone has to be hauled. With the maximum profile gradient of 1:10 an increase of the height of the top bench by 10 m shortens the haulage path by 100 m. Besides lower investments in the roads, their layout is simpler. At the same time haulage costs are less.

An example of how ton-kilometres of broken rock can depend on the vertical division of the quarry face can be given by the following four alternatives for a horizontal deposit of rock with a total area of 10 hectares, occupying an area of approximately 300 m by 300 m. The height of the quarry face is 80 m. Each vertical metre of the deposit represents a mass of rock of 100 000 m^3 volume, i.e. approximately 250 000 tons.* The four alternatives are:

(*a*) The face is divided into four benches, each 20 m high. The benches are broken by curtain blasts. The transport routes from the individual benches to the toe of the face are 0·6 km, 0·4 km and 0·2 km long.

(*b*) Of the three benches, the top and the middle are each 30 m high, with a 20-m bench at the bottom. These are broken by curtain blasts as before. The length of the corresponding transport routes is now 0·5 km and 0·2 km.

(*c*) The top bench, 40 m high, is won by heading blasts, the remainder being divided into two benches each 20 m high, and both are broken by curtain blasts. The transport distances are 0·4 km and 0·2 km.

(*d*) The top bench is of the same height and is blasted as in (*c*) above, the lower half of the face being divided into two benches; the upper, 25 m high, is won by curtain blasts and the lower, 15 m in height, by square blasts. The haulage distances are 0·4 km and 0·15 km.

The total transport effort, that is ton-kilometres, for the four methods is as follows:

(*a*) $(20 \times 0{\cdot}6 + 20 \times 0{\cdot}4 + 20 \times 0{\cdot}2) \times 250\,000 = 5\,500\,000$ t-km

(*b*) $(30 \times 0{\cdot}5 + 30 \times 0{\cdot}2) \times 250\,000 = 5\,250\,000$ t-km

(*c*) $(40 \times 0{\cdot}4 + 20 \times 0{\cdot}2) \times 250\,000 = 5\,000\,000$ t-km

(*d*) $(40 \times 0{\cdot}4 + 25 \times 0{\cdot}15) \times 250\,000 = 4\,937\,500$ t-km

It can be seen that the most advantageous division is to grade the thicknesses of the benches from the surface downwards, so that the lower bench would have the least thickness. Another advantage is that this arrangement makes it possible to apply square blasting in the bottom bench. From the point of view of size distribution of the blasted rock this gives the best results in that part of the deposit where the material is strongest and blocky. Secondary blasting on the crushing floor level is thereby reduced, in a place where it is least welcome.

---

* The metric ton is used throughout this paper.

Summing up, the following arrangement seems to be the most advantageous.

(1) Make the top bench as high as possible and break it with heading blasts.
(2) In the central part use curtain blasting and choose such a height of bench as will give the most economical results in the cost of drilling shotholes.
(3) Break the bottom bench by either curtain or square blasts, depending on the remaining thickness.

COMPUTING THE CHARGES

It follows from the explanation just given that all methods of primary blasting come under consideration. In spite of the present-day tendency towards curtain and square blasts, a total elimination of heading blasts is not intended. This method of blasting is justified in many instances, particularly in cases where the rock breaks readily and where the preparation of suitable conditions for the introduction of curtain blasts would entail a great deal of work. This is commonly the case when opening up new faces on steep slopes where it is not possible to use drilling equipment, etc.

On that account considerable attention has been paid to the technique of heading blasting and to methods of calculating the charges. The latest methods of preparing plans for blasting work show a number of changes from the original conception. Soon after the introduction of heading blasting, the original method of placing the explosive in a small number of charges in a single row was replaced by one in which a larger number of individual charges is placed in two or even three rows. When the explosive is distributed in this manner it is difficult to apply empirical formulae such as were derived by Lares, Weichelt, and others. While such formulae regard the individual charges as producing independently acting forces, in Czechoslovakia a heading blast is looked upon as an ordered system of mutually acting concentrated charges, forming a complex whole.

In simple cases, such as a blast in a free and regular face, the charge can be computed according to the usual formulae. In more complicated conditions, such as development blasts, blasting in tight corners, or blasting faces of a varying height of overburden, such formulae no longer hold, since in such cases consideration of the action of individual charges must be more accurate and worked out in greater detail.

Methods of calculating charges for any kind of a blast can be classified, in principle, into three basic groups.

In the first methods of computing the charges on the conception of specific consumption of explosive are included. The underlying idea is that for breaking a unit volume of rock a certain quite definite quantity of the chosen basic explosive is required. All such methods of computation can be expressed by the relation

$$N = q.V$$

where N = size of charge in units of weight, usually in kg
q = coefficient of proportionality (dimensionless) designated as the specific consumption of an explosive (and as such with the dimension kg/m^3)
V = volume of rock broken by charge N, usually in m^3

All kinds of formulae for the computation of charges which can be reduced to this basic type are naturally empirical, as can be deduced from the fact that, dimensionally, the equation is

$$[\mathrm{kg}] = [\mathrm{m}^3]$$

and the dimensionless coefficient has an apparent dimension $\left[\frac{\mathrm{kg}}{\mathrm{m}^3}\right]$, but only from the relationship $\frac{N}{V} = q$, not from the determination of the coefficient for the various formulae. In spite of their limitations such formulae are still the only ones used in practice. Their basic shortcomings, as well as those of all other empirically expressed relations, are that they are valid only for a limited range of conditions and can lead to false results if applied outside the range of their validity.

The second group of methods for computing charges of explosives is based on the effects the detonating explosive have on the surrounding rock, the course of the disintegration of rock being considered as the effect of the propagation of wave stresses through the rock mass caused by the shock of the detonation wave on the rock enveloping the charge of explosive.

The third group is concerned with the changes of the stresses in the rock and their course and state of deformation, where the instantaneous effect produced by the detonating explosive charge is regarded as one of the possible causes of dynamic action on the rock.

Owing to the complexity of calculations and the difficulties of physical interpretation of the two last named methods, their wider use in current blasting practice is not considered here. At the same time the methods of the second group are not as yet worked out for application in non-homogeneous anisotropic media such as rocks. On the other hand, the methods of the third group are not worked out for cases involving stressing such crystalline structures as rock by instantaneous forces such as are produced by a detonating explosive. Both methods deal with one and the same phenomenon from different angles and it is possible that both approaches will be unified and result in an exact method of computing charges of explosives.

For current practice in determining the charges of mass blasts, particularly heading blasts, the results of both these methods have certain value inasmuch as they make it possible (*a*) to limit the scope within which several of the empirical formulae are valid; (*b*) to carry out the corrections necessary for some of the partial coefficients; and (*c*) to determine the measure of the effect on the blast of some of the magnitudes which are not included in the empirical formulae.

Such an evaluation should be carried out particularly when dealing with complicated heading or chamber blasts. The magnitudes, relationships and characteristics which come under consideration should be dealt with not only qualitatively, but also quantitatively, as they may sometimes be of greater importance than values included in one or the other partial coefficients in the applied formula.

There are many factors entering into the breaking of rock by explosives. The final result of a blast depends not only on the character of the explosive but also on the mechanical properties of the rock.

First, it is necessary to determine the rock's characteristics from the point of view of its mechanical and physical properties. For exact treatment it would be necessary to consider that the entire block which is separated from the coherent body of rock is a continuous connected medium in which all the changes take place simultaneously and with equal intensity. Such a condition can be found, if only partially, in such special cases as in driving drifts in compact igneous rock and similar media. Blasting operations in free quarry faces take place in a mass of rock with parting planes of various sizes and directions which give the rock the character of a discontinuous medium. If the parting planes are not in the form of cracks filled with material of quite different mechanical and physical properties, such as clay, loam or similar material, and if the surfaces of the parting planes are in intimate contact, the effect of discontinuity is not so marked as to preclude the application of mathematical methods, albeit with certain limitations.

The existence of parting planes inside the block to be broken determines the ability to deform the state in which the rock may be in such an environment. The rock is in an elastic or brittle condition, not plastic or deformed. In principle, it means that all phenomena which take place inside the block after the detonation of the charge are within the limits of validity of Hooke's law, where the relationship between stress and strain is constant and is expressed as the modulus of elasticity (Young's modulus). While it is sometimes affirmed that blasting can cause plastic deformation, this is to be understood as meaning that what takes place is the destruction of the entire crystalline structure, not a non-destructive plastic deformation which is permanent and measurable, a deformation of volume and shape.

The elastic deformation which takes place is propagated with the velocity of sound in the entire system, where it causes stresses of similar magnitude. With very high velocities of loading the deformation in the mass, particularly in a mass of crystalline structure, is only elastic. It becomes plastic after the elastic limit has been exceeded; the stress at the limit of plasticity is at the same time the stress at which failure takes place. In contrast, the deformation when ultimate strength is reached is always greater than the deformation at the limit of plasticity. Only in brittle materials is it possible that deformation at the limit of plasticity and when the ultimate strength is reached is the same. That is why such materials are strained only elastically and plastic deformations do not take place before failure. Only in special cases can plastic deformation, in its true sense, occur, i.e. where permanent deformation of volume and shape takes place, for instance in compacting embankments in plastic earths, in joints of rock filled with clay, etc., by explosives.

In principle, these considerations are valid only for homogeneous systems. In non-homogeneous systems a combination of both elastic and plastic deformation can take place—for instance when crystals of greater strength are deformed within their elastic limit, while crystals of lower strength become deformed plastically. A deformation both plastic and elastic can take place in a rock mass if the charge of explosive is weak and a disintegration of the body does not take place. The same happens where the block to be broken is highly stressed from all directions. Owing to lateral pressure, brittle crystalline matter changes to elasto-plastic material, depending on the rate of growth of this pressure. Theoretically it is possible to attain an infinitely high strength of any material if it is contained by a uniform pressure from all directions. Such a substance remains perfectly elastic. A uniform side pressure diminishes or eliminates lateral tension of bodies under compression. Lateral failure of the body is thereby limited and its strength is increased. At the same time the angle which the axis of the body encloses with the sliding plane when shearing by pressure, as well as the magnitude of the limiting stress, are increased. Thus, for instance, the angle of the sliding plane of marble which amounted to 26°30′ at zero side pressure increased to 45° under high side pressure.

The effects of high side pressure are particularly noticeable when driving drifts and tunnels and in heading blasts when one of the charges is placed in the corner of a face and the charge acts against a vaulted surface. A case of this kind, by no means rare, is very sensitive to a correct computation of the charge. On one hand, too low a charge may be estimated and the block remains 'sitting', as after a miscarried blast. (This often happens when the time interval for the initiation of the neighbouring charge was wrongly chosen and the marginal charge is doing its work under pressure from both sides.) If, on the other hand, the effect of the side pressure is overestimated, the throw of the blasted material is excessive and may damage quarry equipment.

The prerequisite of a successful blast is a sufficiently accurate estimation of all the factors which influence the blast.

Of the mechanical values which are of significance for judging the course of the process of disintegration, besides the stress σ as a common comparative value, the principal is the strain or deformation ε. The degree of deformation and its character depend on the rate of change of the strain $\left(\frac{\partial\varepsilon}{\partial t}\right)$ which is the function of the mode of loading and the speed of loading, as well as on the magnitude of the stress, its time rate, and the gradient of the tension in the rock. When loading the rock at a high rate, elastic deformation takes place and the fragility of the material increases. An important factor for judging the result of a blast is the rate of change of the deformation $\left(\frac{\partial^2\varepsilon}{\partial t^2}\right)$ which assumes very high values at the instantaneous working of the charge and at alternating directions of the deformation, that is when tension alternates with compression. Under such conditions the strength of the body is progressively decreased by the formation of fissures perpendicular to the direction of compression and increased by the ensuing tension.

If the system is unable to spread the stress uniformly, for instance, on account of its crystalline structure, then such a gradient of tension in the rock results $\left(\frac{\partial\sigma}{\partial r} = \text{the increment of stress } \Delta\sigma \text{ in a length } \Delta r \text{ approaching zero}\right)$ that it exceeds the limiting values and brings about such a relative displacement of the elements of the system that its complete disintegration takes place. That is why substances of higher strength but less plasticity can fail due to a high gradient of tension sooner than substances of a lower strength but capable of greater deformation (compare, for instance, diorites and tuffs in this respect).

Besides these factors the initial conditions, which may be quite varied, also play their part in the course of the blast. It is, for instance, the effect of prestressed masses of rock which is useful in some cases of corrective blasts in unstable faces, or where there is a variable thickness of burden over the charge, etc.

In any given case of a chamber blast it is necessary to take into consideration the basic characteristics of the block as a whole. Principally, the rock has the character of a non-homogeneous anisotropic medium. Lack of homogeneity must be respected, particularly in cases where the mechanical and physical characteristics of the individual sections of the line of least resistance abruptly change. Such a case may be found in the correction of an unsuccessful blast where the charge is put into sound rock and the line of least resistance consists of one section in sound rock and another in disintegrated rock. In such a case the burden is equated to a new magnitude of a physically homogeneous burden. A similar course is taken when there is a marked heterogeneity in the character of the overburden (earth or weathered rock) or in the strata above the charge in development blasts.

Anisotropy must be taken into account particularly when dealing with sedimentary rocks, some types of metamorphic and altered igneous rocks where mechanical and physical properties vary according to the direction in which they are stressed. In many cases, however, the anisotropic properties of the mass balance, and the rock as a whole behaves like an isotropic mineral. The higher the number of balancing elements, the higher is the degree of isotropy. The effect of anisotropy decreases with an increase in the size of the block and therefore also with the size of the charge. Anisotropy is felt, however, when using small charges and when the blast does not cause complete disintegration of the rock. Anisotropy must be taken into account when the direction along which the lower limiting values of the individual magnitudes act is orientated towards the mechanical equipment of the quarry which might be endangered by the throw of the disintegrated rock. Such a situation may arise in connection with sedimentary and metamorphic rock with inclined strata.

Blasts are in general classified into those causing shock, loosening, breaking and throw of rock. It is obvious that in each type of operation, when there is no concern with the effect on the immediate environment, the effects will be other than those designed. In blasts where the charge is used to produce shock, for example, the main purpose is to achieve elastic deformation without reaching limiting values. In blasts for loosening strata, the principal aim is to produce permanent deformation of the medium, i.e. to exceed the limiting values in the realm of elastic deformation, but not to exceed the stage of permanent plastic deformation (as when compacting embankments, loosening earth, etc.). Ripping and blasts to throw or lift the rock represent blasts in which all the limiting values of deformation are exceeded, even if the structure of the system is vitiated when the limits of elastic deformation are surpassed.

Such considerations are of significance when the parameters of the blast do not fall within the validity of empirical formulae, which were derived under actual conditions and for a given practical application.

Methods for computing charges for industrial applications originally placed reliance on formulae used by military engineers. However, as the deciding factor in military applications is to make the result absolutely sure, for industrial uses of explosives it is necessary to balance economy with the probability of attaining the desired objective without jeopardizing the safety of the immediate environs of the locality where blasting is being done. The majority of formulae for computing charges for civilian use being based on military formulae, these are supplemented and corrected by various coefficients, the object of which is to secure the maximum economic effect with the minimum quantity of explosive.

All empirical formulae can be expressed by one relation, i.e. linear dependence between the weight of explosive and the volume of rock to be disintegrated, where the constant of proportionality is the so-called specific consumption of the explosive. This relationship was first formulated by Vauban, i.e. that the charges are proportional to the third power of the burden (w)

$$N:N_1 = w^3:{w_1}^3$$

where

$$\frac{N_1}{{w_1}^3} = q$$

is the specific consumption of the explosive, it being assumed that the charge will separate such a cone of rock whose radius (r) is equal to the burden (w), i.e. $r = w$, and the angle of the cone is 90°. Then the volume (V) of this cone is

$$V = \frac{\pi r^2 w}{3} = \frac{3{\cdot}14\, w^3}{3} = w^3$$

This expresses the proportionality between the charge and the volume of disintegrated and lifted rock and the formula is

$$N = q\,.\,w^3$$

This is also known as Hauser's formula, with the difference that Hauser introduced a further coefficient (t) expressing the influence of sealing, so that the formula is

$$N = w^3.q.t$$

Values for $w > 2$ m were worked out by Stobbe on the basis of results from blasts actually conducted. For blasts which are to break the rock, but not lift it, the charge is reduced by multiplying it by coefficient $\frac{1}{3}$ up to $\frac{1}{8}$. With an average value of $\frac{1}{5}$ the charges according to the Hauser–Stobbe formula give values which, where the burden exceeds 10 m, approximate the values of Lares' correction.

The improvement introduced by Lares consists of establishing the fact that in winning minerals it is not necessary to dimension the charge so as to lift the rock from a right-angled crater, but that it will suffice if the ratio (n) of the radius (r) to the burden (w) will be less than 1. Lares has further introduced a variable coefficient for expressing the relationship between the military and civilian charges. The coefficient is $f(n) < 1$ at burden $w > 1$ m.

The same worker also prepared a global coefficient (q) expressing the specific consumption as a product of several partial tabulated coefficients, the well known expression for q being

$$q = f.s_1.s_2.v.\frac{e}{\Delta}.d$$

A detailed study of Lares' formula was carried out by Weichelt. His corrections and additions consist of expressing the fact that the validity of Lares' formula is not absolute and that for every application a certain additional correction must be made. In this way Weichelt obtained an entire series of formulae for various conditions, particularly for various ratios of burden (w) to the height of the face (H) and for various distances between the charges.

It can be said that all empirical formulae are based on the same principle formulated by Vauban, with the distinction that a correction coefficient is introduced for the use of explosives when winning minerals. This coefficient $f(n)$ amends the basic (military) charge which can be called the efficiency coefficient. This coefficient, worked out by various authors (Lares, Ohnesorge, Meinecke, Frolow, Boreskov, Humpertz-Lebrun) is expressed in various ways and derived by different methods, but in principle it is the same coefficient.

The values of q also vary with different investigators and are usually tabulated as a summary coefficient or are detailed in the form of further partial coefficients. The differences between the values of these coefficients worked out by various authors are undoubtedly due to different geological conditions under which they were ascertained and depend on the types of explosives used locally and on the strength of the standard explosive used for comparison. It has been shown, for example, that the coefficients formulated by Lares and Weichelt for certain rocks cannot be used in their entirety under different geological conditions in Czechoslovakia.

A somewhat different approach was followed by Jurajda in preparing a formula; although this also was based on the specific consumption of the explosive, he used a different method of deriving the various coefficients. The choice of the correct coefficients does not require long experience, all values being given by geometrical and mechanical magnitudes and relations which are simple and easy to ascertain, as is evident from the formula whose expression for computing a concentrated charge is

$$N = k(w^3 + w^2)c$$

where c is the global coefficient including various magnitudes which influence the course and the result of the blast. Coefficient c is determined from the relationship

$$c = (p + \gamma).\frac{b}{h} u.\alpha.g.t$$

where N = the size of the charge in kg
k = efficiency coefficient of the charge
w = burden of the charge in m
p = the strength of the rock in tons/cm²
γ = specific weight of the rock in tons/m³
b = coefficient of the strength of the explosive
h = loading compactness of the charge
u = coefficient of the stability of the block to be blasted
α = coefficient of co-operation between charges
g = coefficient expressing the effect of gravity on the main direction of the blast
t = coefficient of sealing

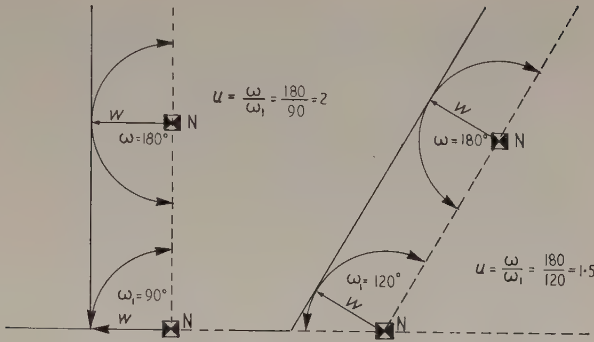
It is clear that all these values can be ascertained objectively or measured and are not dependent on long experience. The magnitude of the coefficients is calculated from known measured values. The coefficient for the stability of the block is taken as the ratio of the volume of an ideal hemisphere of efficiency for a blast acting only towards a free face to the real spherical sector (cone) under the given conditions. This coefficient can therefore be expressed

$$u = \frac{V_i}{V_1}$$

If the values differ only in one plane then this relationship can be expressed as the ratio of vertex angles and, as is evident from Fig. 2,

$$u = \frac{180^\circ}{\omega_1^\circ}$$

Fig. 2.


$u = \frac{\omega}{\omega_1} = \frac{180}{90} = 2$
w
N
ω=180°
ω₁=90°
w
N
w
ω=180°
N
$u = \frac{\omega}{\omega_1} = \frac{180}{120} = 1{\cdot}5$
ω₁= 120°
w
N

The coefficient of the co-operation of charges (α) is given by the equation

$$\alpha = \frac{(L/2 + w)^2}{2w^2}$$

where L is the distance in metres between the charges. In practical blasting the value of this coefficient is taken to equal 1.

To satisfy the previous condition for the value of α the distance between the charges must satisfy the equation

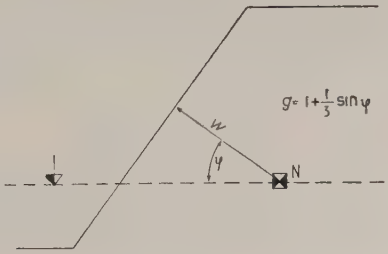
$$L = 0{\cdot}83\, w$$

Coefficient g, taking into effect the gravitational force, expresses the increased consumption of energy for lifting the disintegrated rock against gravity. This coefficient is determined from the relation

$$g = 1 + \frac{\sin \varphi}{3}$$

where φ is the angle which a horizontal plane encloses with the direction of the blast (Fig. 3).

Fig. 3.



If all these formulae arrived at empirically are compared, certain elements limiting their application will be found. In all formulae there is a coefficient expressing (in different forms) a dependence of the quantity of the charge on the strength of the mineral (Lares–Weichelt $= f$; Jurajda $= p$). In all cases the strength of the rock is taken as the strength in simple biaxial compression, regarded as a constant characteristic of the mineral. However, the compressive strength is a variable quantity growing with the magnitude of side pressure which cannot be disregarded in blasting. At the same time the fragility of the material rises under instantaneous loading and the region of plasticity is not apparent. In a free face it can safely be assumed that such effects are small and that the value expressing the strength of the rock as applied in the formula will be satisfactory. With high stability it is necessary to take these effects into consideration and to estimate to what degree they make themselves felt. The formulae also contain a coefficient which provides for the effect of sealing the charge in the compact rock. The effect of side pressure is included in the form of increased resistance against shearing. Strength in shear is closely connected with the strength of the material, but is not identical with it.

The formulae further assume a linear relationship between the strength of the rock and the size of the charge. This is quite satisfactory in the range of medium values of strength, but not in the case of high or low values. With basalt, which has a high strength in compression but is quite cleavable, the calculated charges are too high. Tuffs are an opposite case, where it is necessary to increase the charges resulting from the computation. Mere characteristics of strength do not suffice in cases of extreme values and it is necessary to take into consideration other factors entering into the disintegration of rock. That makes the coefficient dependent on several values and the current method of determination approximates reality only in the range of average values. The same is true of the global coefficient of Jurajda's formula (items p and γ).

Such limitations of validity affect calculations for heading blasts. Most authors are aware of the complexity of planning blasts of this type and have formulated rules and parameters which have to be adhered to when proposing such a blast. In most cases they agree on the condition for the burden (w) and the height of the face (H) and that the conditions are satisfactory if they comply with the condition

$$\frac{H}{3} < w < \frac{2H}{3}$$

These limits are also the boundaries within which the formulae are unconditionally valid. If not it is necessary to make corrections for factors not included in the formula. This is the case not only with very low faces where $w > \frac{2H}{3}$, but also in high faces where, from practical considerations (burden too large) it is not possible even to keep w equal to $\frac{H}{3}$ or where a good parting is precluded owing to the danger from overhanging rock (as is the case with some basalts and limestones).

In low faces where $w > \frac{2H}{3}$ the block is lifted before falling and a heap of disintegrated rock is piled up to the rear. In the course of such a blast a certain loss of energy contained in the explosive takes place to overcome the effect of gravity and the material is not moved sufficiently from the face. If the same result and throw of the shattered rock is desired as with faces of medium height, it is necessary to increase the charge over and above the figure given by the formula. However, in exceptional cases, as for a charge in a low wall, the reduced value should be increased by no more than one-third to cover the effect of gravity. With high walls, where the burden is less than one-third of H, values given by current formulae are too high, as indicated by the spreading of the throw. The calculation of charges, siting and behaviour in multi-row heading blasts used to prepare new benches, where part of the force of explosion acts towards the overburden and part towards the base of the quarry, is somewhat complicated. A similar situation arises when blasting ridges between two faces, where charges act in opposing directions.

In all such cases the current methods of computing charges are valid only to a limited extent. When planning such blasts considerable care must be exercised; the entire course of the changes in the deformation of the rock must be studied and on that basis the scope of usefulness of the formula be determined and the influence of other factors which are not taken care of in the formula assessed.

A distinct advantage of the Jurajda formula is the fact that with further partial coefficients it is possible to cover a relatively large number of cases. His formula is suitable, for instance, for development blasts on slopes and it is also applicable to conversions of calculations for curtain or deep hole blasts to determine charges for square blasts. Only on rare occasions are blasts carried out under conditions where the validity of this formula is limited and must be supplemented by further considerations and computations.

CORRECTIVE BLASTS

With the increase of capacities at quarry faces, it is necessary on occasion to carry out corrective blasts in a face vitiated by a poor blast or to correct the results of a blast which did not disintegrate the rock sufficiently. The principle of all such blasts is to place the charge in sound virgin rock for (1) better control of the action of charge, and (2) safety of operations when drifting or tunnelling. In contrast, a drive in affected strata represents increased costs of timbering and the ever present danger of caving. A disadvantage is that the charge acts in two quite distinct environments. This lack of homogeneity has often been the cause of miscarried blasts.

When calculating charges for such conditions, a fairly correct estimation must be made as to the location of the boundary between sound rock and shattered rock. The resistance line will consist of two segments, each with quite different characteristics. When computing the charge this circumstance must be neutralized in a way so that, shifting the shattered rock with disintegration and throw of the sound rock can be achieved.

There are several ways of doing this. Theoretically, the best would be to regard the locality as two independent cases and to compute a charge for each of them. However, this approach is far too complicated as it presupposes a calculation based on power impulses and the propagation of lateral waves of tension and compression with a passage of energy from one environment to another.

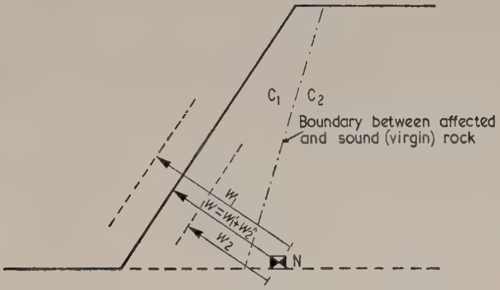
The second possibility is to carry out the computation with the help of the usual formulae. In such a case it will be necessary to correct some of the coefficients to a value corresponding to the heterogeneity of the environment. This method does not entail much work, but does include the danger of choosing incorrect coefficients, particularly if the position of the boundary between sound and shattered rock is wrongly estimated.

The third and simplest way consists of determining the upper and the lower limit of the size of the charge by approximating the values of the charges within the extreme limits of accuracy afforded by the empirical formulae. In this way the magnitude of the resistance line and the boundary plane between the two regions are determined. To that end the quarry face is measured and the changes encountered in driving the approach adit for the new charge will indicate the probable location of the boundary plane. It will be easier if the adit can be driven through impaired rock, but more difficult in virgin ground. In that case the run of the boundary plane is more difficult to locate, but its approximate location may be revealed by evidence on the face. This, however, requires experience and a knowledge of geology.

After determining the location of the boundary plane in the environment of the charge, the measured burden (*w*) is recalculated to the length for computing the charge. This adjustment is done on the basis of proportionality of the physical and mechanical properties of the medii. It is considered that the shattered rock must only be shifted while the virgin rock must be broken as well. When placing the charge in sound rock it is not necessary to take into account the loss of energy due to the gases of explosion escaping into free spaces. It will be noted that the global coefficient of the empirical formulae (c) for the affected rock is always lower than the equivalent coefficient for virgin rock. The coefficients for both regions (c_1 and c_2) are first determined, and as it is easier to determine the coefficient for sound rock than for affected rock a greater accuracy is attained if coefficient c_2 is recalculated from coefficient c_1.

After the coefficients and their respective segments on the resistance line have been settled, the heterogeneous burden is recalculated to a homogeneous burden of the same physical and mechanical characteristics requiring the same amount of energy. The burden will either be greater than the actual burden if based on coefficient c_1 of the affected rock, or smaller if based on the coefficient of sound rock c_2. These relations will be better understood by reference to Fig. 4, which shows that the actual burden w consists of segments w_1' and w_2'.

Fig. 4.



As it is known that c_1 is smaller than c_2, the individual charges N_1 and N_2 would be proportional to the third power of their respective burdens, i.e. $N_1 = w^3 . c_1$ and $w^3 . c_2 = N_2$. The actual charge N will therefore lie between the two values so that $N_1 < N < N_2$ because $w^3 . c_1 < w^3 . c < w^3 . c_2$, from which $c_1 < c < c_2$ follows.

From this relationship an approximate, though rather inaccurate, value of the equivalent coefficient c and from it also that of the charge N can be ascertained, i.e. $N = w^3 . c$, where $c = \dfrac{w_1 c_1 + w_2 c_2}{w}$.

This value will suffice if the difference between c_1 and c_2 is small, within the limits $c_2 - c_1 \leqslant 0{\cdot}2\, c_2$. If the difference is larger than $0{\cdot}2\, c$, the approximate value will serve in considering whether it will be necessary to carry out an approximation of the first or the second degree.

Otherwise it is necessary to recalculate the values for a burden which will have the same characteristics throughout its entire length and will be equivalent to the burden ascertained by actual measurement. For $c_1 < c_2$ the equivalent segment will be

$$w_2' c_2 = (w_2' + d) c_1$$

or

$$w_1' c_1 = (w_1' - d') c_2$$

i.e. the total burdens will be in the relation

$$w_2 < w < w_1$$

As the charges corresponding to the burdens w_1 and w_2 will be the same, i.e. $N_1 = N_2$, then

$$w_1^3 . c_1 \text{ must be equal to } w_2^3 . c_2$$

Expressing the burdens w_1 and w_2 as dependent functions of c_1 and c_2, then

$$w_1(c_1) = w'(c_1) + x_2(c_1)$$

or

$$w_2(c_2) = w_2'(c_2) + x_1(c_2)$$

where

$$x_2(c_1) = w_2' . (c_1 c_2 + c_2) = w_2' c_2 (c_1 + 1)$$

and

$$x_1(c_2) = w_1'(c_1 c_2 + c_1) = w_1' c_1 (c_2 + 1)$$

then

$$w_1 = w_2' c_2 (c_1 + 1) + w_1'$$

$$w_2 = w_1' c_1 (c_2 + 1) + w_2'$$

and the charge

$$N_1 = w_1^3 c_1 = [w_2' c_2 (c_1 + 1) + w_1']^3 . c_1$$

$$N_2 = w_2^3 c_2 = [w_1' c_1 (c_2 + 1) + w_2']^3 . c_2$$

In an actual example where, for instance the burden $w = 15$ m is composed from a segment of affected rock where $w_1' = 11$ m with coefficient $c_1 = 0{\cdot}5$ and a segment of virgin rock where $w_2' = 4$ m with coefficient $c_2 = 0{\cdot}8$ the charges will be

$$N_1 = 15^3 . 0{\cdot}5 = 3375 . 0{\cdot}5 = 1700 \text{ kg}$$

$$N_2 = 15^3 . 0{\cdot}8 = 3375 . 0{\cdot}8 = 2700 \text{ kg}$$

These are the extreme values of the charge. For an average coefficient

$$c = \frac{11 \times 0{\cdot}5 + 4 \times 0{\cdot}8}{15} = 0{\cdot}58$$

gives the value of the charge $N = 1960$ kg.

When computing the charge by approximations, the values for the length of the burden and the size of the charge are as follows:

$$w_1 = 4 \times 0{\cdot}8 \times (0{\cdot}5 + 1) + 11 = 4{\cdot}8 + 11 = 15{\cdot}8 \text{ m}$$

$$N_1 = 15{\cdot}8^3 \times 0{\cdot}5 = 2000 \text{ kg}$$

and $$w_2 = 11 \times 0{\cdot}5 \times (0{\cdot}8 + 1) + 4 = 13{\cdot}90 \text{ m}$$

$$N_2 = 13{\cdot}9^3 \times 0{\cdot}8 = 2150 \text{ kg}$$

The difference between the values of the charge N_1 and N_2 is about 7 per cent, which corresponds to the precision of results when using empirical formulae and the limits thus ascertained can be regarded as satisfactory. This method of calculation, though approximate in character, has the advantage of simplicity.

An exact computation based on the relationship of volumes and sliding planes of both medii and the requisite power impulse would necessitate a complicated calculation involving the application of differential equations, a method hardly suitable for everyday quarry practice.

A method based on the relationship between volumes of the different medii is simpler but complicated by the fact that the boundary plane is usually not perpendicular to the axis of the main direction of the detonation.

CONCLUSION

Some facts accumulated in the process of introducing large-scale production in stone quarries have been presented here. A study of the theoretical and practical aspects dealing with the dimensioning of charges and the application of sufficiently accurate methods of computation, particularly the formula proposed by Jurajda, and their application, has made it possible to introduce mass production methods of quarrying in a relatively short time and without high capital investment, even in quarries where buildings, etc., liable to damage by blasting are close by. In a number of instances chamber blasts have been carried out within 20–30 m of the stone preparation plant, road or railway. Several sufficiently accurate blasting methods are available from which that best suited to the character of the particular quarry face can be chosen, on the basis of experience gained, to give the best results for mass production.

622.271.2:622.355.11(410)

The Limestone Quarrying Operations of Imperial Chemical Industries, Limited, Great Britain

R. G. JACKSON, M.Eng., A.M.I.C.E., A.M.I.Min.E., M.I.Q.
Operations and Works Manager, Buxton Lime Group, Imperial Chemical Industries, Limited, Mond Division

---

SYNOPSIS

Large tonnages of limestone are won from the I.C.I. limestone quarries each year for use in chemical and other processes.

In the paper historical reference is made to I.C.I.'s interest in quarrying, and some of the considerations leading up to the choice of a particular technique or piece of equipment are discussed. Mention is made of the crushing, screening and scrubbing processes, which are regarded as an integral part of the quarrying operation. The achievement of a generally progressive reduction in the accident frequency rate (0·38 in 1962) is noted and commented on. Supervisory arrangements are also briefly described.

The limestone quarries of I.C.I. produce large tonnages of crushed and graded limestone for use in chemical processes and for conversion into a variety of lime and limestone products.

In all, five quarries are at present being worked, and the Company's experience in quarrying ranges from the working of a quarry, which by world standards is large, down to the smaller quarries more frequently encountered in this country. In direct association I.C.I. also operates vertical kilns, rotary kilns, hydrating plants and a number of ground limestone plants. The most important producing centre is at Tunstead, near Buxton, in Derbyshire, where, in addition to the quarry, high-capacity vertical kilns rotary kilns, and a large hydrating plant are in regular production.

While the present paper contains matters of interest from all five quarries, greater reference is made to the Tunstead operation, where much quarry investigatory work has been done. It is not considered of interest to embark on descriptions of standard pieces of equipment, or to make more than passing reference to methods standard throughout the quarrying industry, but an attempt has been made to concentrate attention on unusual features or experiences.

HISTORICAL

The interest of I.C.I., or its predecessors, in limestone quarrying goes back to 1872, when John Brunner and Ludwig Mond introduced the alkali industry at Northwich in Cheshire and started working the Solvay process the following year. The siting of their works was undoubtedly influenced by the plentiful supplies of high calcium limestone available some 35 miles away by rail in the Buxton area.

At that particular time, limestone was being produced by a large number of fiercely competitive small firms. No one company made a big profit and it was easy to make a loss. Fortunately, however, several of the small companies directors soon appreciated the position and in 1891 thirteen of the surviving companies amalgamated to form the Buxton Lime Firms Company, which by 1926 consisted of 22 works and included some 55 vertical kilns of varying standards of efficiency, two Hoffman kilns, seven crushing plants, two hydrating plants and six quarries operating solely for the direct sale of stone.

During this time John Brunner and Ludwig Mond (by now Brunner Mond & Company) had been taking a considerable proportion of the limestone production of Buxton Lime Firms, and in 1918 their dependence on this limestone had grown so great that they bought a controlling interest to safeguard their source of supply. In 1926 Brunner Mond, together with the Buxton Lime Firms, were party to the great merger which led to the formation of Imperial Chemical Industries.

In 1926, as a result of the formation of I.C.I., Raynes quarry at Llysfaen, on the North Wales coast, was taken over, its principal function being to supply the ammonia soda works at Fleetwood with limestone. The latest acquisition—in 1961—has been that of Settle Limes, Limited, whose works are in North Yorkshire.

At present the combined quarries and plant at the three producing centres, i.e. at Buxton, North Wales and Settle, form part of one of the I.C.I. Divisions. Production is under the control of one works manager, who is responsible to a Division Product Group Director.

From the time of the I.C.I. merger it became the object of the Company to concentrate its production on a few highly mechanized and efficient quarries. One by one the older and smaller quarries were closed and, parallel with this, a large new quarry at Tunstead was developed (Fig. 1). It is interesting to note that since the beginning of the Second World War the number of men on the payroll at Buxton and at Raynes quarry, both for quarrying and other operations, has dropped from about 2000 in 1939 to 750 in 1962, a decrease of approximately 60 per cent. The annual tonnage of stone quarried during this period has increased by 45 per cent, and there have been very substantial increases in the output of the other main products. There has also been a distinct change in the uses to which the large tonnages of stone are put, as

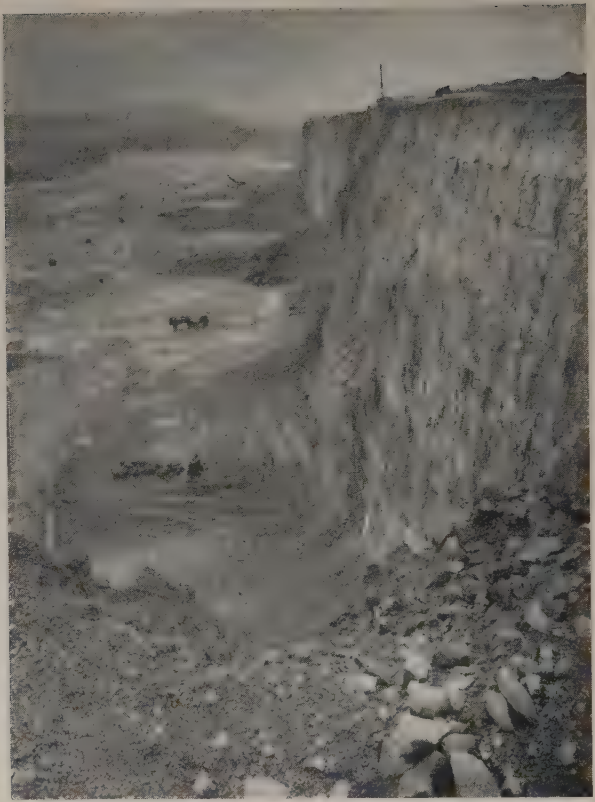


Fig. 1.—North end, Tunstead quarry (photograph taken in 1955).

a result of the vast expansion that has taken place in lime, hydrate, aggregates and other products.

Before the end of the Second World War most of the quarrying was carried out by the use of well established and trustworthy processes. Primary blasting, for example, took place by driving headings into the quarry face and chambering out—a technique which gave poor fragmentation and resulted in very high secondary blasting costs. Further, there was at that time no known method of producing a clean limestone for chemical use from clay-contaminated crusher feed, and the cleaning process was by selective hand-loading. In fact, an attempt at mechanizing one of the quarries in 1918 failed, owing to the lack of an economical method of cleaning the product efficiently, subsequent to shovel-loading in dirty conditions. The shovels had a 4-yd^3 dipper and were steam operated, loading into broad-gauge wagons which fed a 48-in by 72-in jaw crusher.

Developments towards the end of the Second World War, however, enabled this difficulty to be overcome and the era of modern quarrying began. Labour and economic considerations limited the rate at which mechanization could take place, and over ten years elapsed before the quarries at Tunstead and Hindlow in the Derbyshire area and at Llysfaen in North Wales were all mechanized.

A great deal of work has been done since mechanization was first introduced, and the period subsequent to the introduction of the first mechanical excavators and the commissioning of modern crushing plants has been one of constant development. Reference has already been made to the large increase in productivity which has taken place since 1939. In addition, significant improvements have been made in the recovery of profitable materials from the crusher feed. The present figure is about 95 per cent, and conversion of the remaining 5 per cent waste into a usable product is already under consideration.

GEOLOGY*

Limestone of one sort or another in Great Britain is found to some extent in nearly all the geological systems, but it reaches its maximum development in the Carboniferous, where the quarries account for nearly half the national output. The Pennines are mainly Carboniferous and it is here that the principal lime-producing centres of the Midlands and North are found. In early Carboniferous times the greater part of Britain was covered by sea, with the exception of an island belt running from mid-Wales across England to the Wash, known geologically as St. George's Land. The duration of the Carboniferous system, which includes the Carboniferous Limestone, the Millstone Grit Series and the Coal Measures, was approximately 55 million years, i.e. from 275 to 220 million years ago. Subsequent to deposition, crustal movements and erosion resulted in the limestone outcrops being exposed in the positions they occupy at the present day. The fractured edges of the Millstone Grit and, beyond these, the Coal Measures, can be seen in E–W cross-sections.

In 1947 I.C.I., Ltd., put down an exploratory bore-hole at Woodale, near Buxton, on the axis of the limestone anticline. The base of the limestone was encountered at 892 ft below the surface, resting on rock believed to be Precambrian. As a limestone succession of at least 1500 ft is exposed in North Derbyshire, it appears that the limestone thickness to its base in this area is at least 2400 ft. In both the Settle and Colwyn Bay areas there is also evidence of ancient rocks at the base of the limestone.

---

* EDWARDS, W., and TROTTER, F. M. The Pennines and adjacent areas (3rd ed.). *Bri. reg. Geol.* (London: H.M.S.O. for Geol. Surv. and Museum, 1954), 86 p.

SMITH, B., and GEORGE, T. N. North Wales (3rd ed.). *Bri. reg. Geol.* (London: H.M.S.O. for Geol. Surv. and Museum, 1961), 96 p.

Locally, there are several features which influence quarrying methods and layouts; thus in the Buxton area individual beds are massive, sometimes reaching a thickness of 30 ft. This is in contrast to North Wales where the beds average only a few feet in thickness, and fragmentation is a relatively simple problem. The beds are separated by nearly continuous horizontal bedding planes containing thin bands of clay of volcanic origin, varying in thickness from a fraction of an inch to several inches and, in exceptional cases, nearly a foot. These clays are frequently colour banded. There are, in addition to the horizontal bedding planes, two sets of vertical joints at right angles, those running parallel to the working face being referred to as 'backs'. It is these clay-filled joints which originally dictated the necessity for developing an efficient cleaning process before mechanization of the quarries could begin. There is no clay problem in the Settle area.

At Tunstead, where the face is 1¼ miles in length, the main continuity of the strata is broken by occasional faults, but fortunately the displacement in this part of Derbyshire rarely, if ever, exceeds 100 ft, so that the succession is practically continuous throughout the quarry. On occasions there is, however, local complex faulting which, from a safety angle, has necessitated special quarrying methods. It is in these shattered and displaced rocks that mineral-containing solutions have deposited barytes and galena.

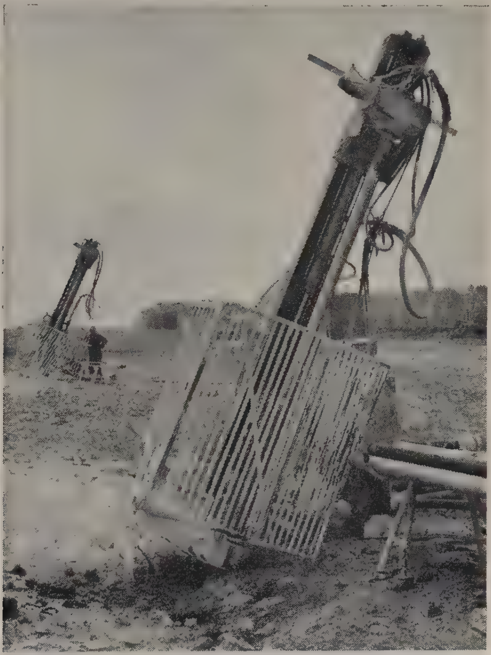
The Tunstead limestone has an average crushing strength of 24 000 lb/in^2 and that at Raynes quarry 22 000.

DRILLING

Drilling and the associated technique of blasting have such a pronounced effect in quarrying operations that in 1959 the company decided to carry out a series of drilling tests with the object of defining more clearly the optimum conditions at their largest quarry. Investigations lasted about two years and final results were based on the average of a large number of drilled sites and primary blasts. They involved trials on nine different types of drill, results confirming that with conventional high explosives, for given fragmentation conditions, the quantity used diminished substantially as the diameter of hole lessened. Using small-diameter high-speed drills there was a saving in drilling costs both over the old churn drills, which were the sole means of primary drilling until 1958, and the larger-diameter rotary drills which were then available for trial. This conclusion was reached in the face of strong opposition from the large-diameter drill manufacturers and, indeed, appeared to be against the general trend in the United States which at the time seemed to be towards an increase from 9-in to 12-in diameter holes.

Two other factors, however, influenced the final decision on the choice of drill and diameter of hole—the advent of ammonium nitrate as a blasting agent and the desirability of drilling holes at an angle to the vertical. At the time no really free-flowing ammonium nitrate was available in this country and the choice was confined to a commercial crystalline product treated with an anti-setting agent. As was expected, there was a distinct tendency for bridging to take place with small-diameter holes and experiments revealed that the smallest hole which could be used with confidence with angles up to 25° from the vertical, using this very cheap blasting agent, was $4\frac{3}{4}$ in. in diameter.

Fig. 2.—Angle-hole drilling at Tunstead.



So far as drilling off-vertical was concerned, two high-speed rotary drills had been purchased, but difficulty was experienced in drilling downwards to a straight line and detailed surveys revealed that the bottom of a hole drilled at even modest penetration rates could deviate by as much as 12 ft in a 180-ft face. Despite almost indignant assurances to the contrary, a number of quarry faces have since been examined and have shown that I.C.I. was not alone in this experience. On the other hand, the Company's experiences with drills using a down-the-hole hammer show clearly that a straight hole could be drilled with reasonable confidence and this observation defined the type of drill necessary to meet requirements (Fig. 2).

At the same time it was appreciated that a large volume of new models of drill would be brought out in the subsequent few years and in working out the economics of providing new drills it was stipulated that these should be written off in five years. Further, experience with tractor-mounted drills operating on a very rough quarry top had shown them to be out of operation for lengthy periods because of track maintenance. This point may well cease to be important in the future, in view of the fact that the method of stripping now leaves a cushion of a few inches of topsoil between the actual rock surface and the drill.

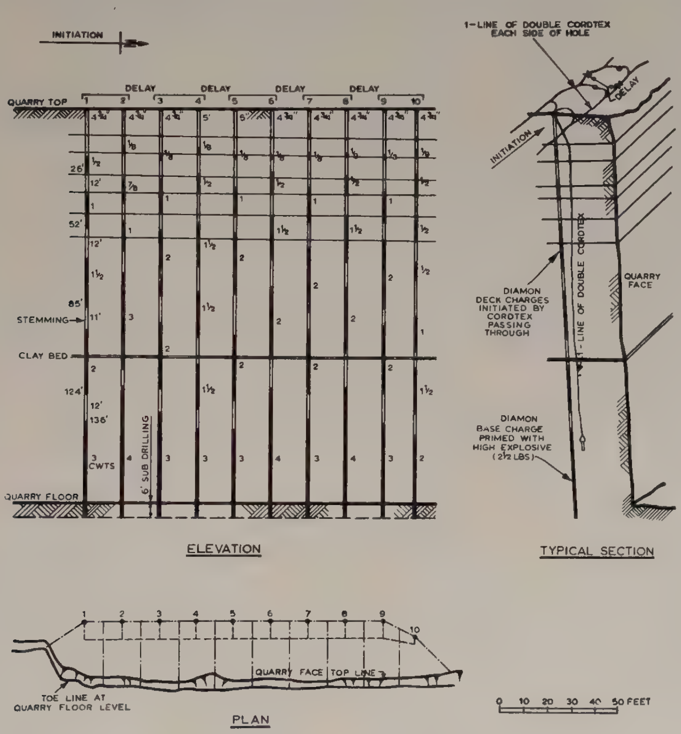
This specification considerably narrowed the choice at the time, but eventually a drill rig was found which, when modified, approximated to these requirements. This gave an average penetration rate of 16 ft/h on a $4\frac{3}{4}$-in diameter hole throughout the working day and a penetration rate during actual drilling of 25 ft/h, which corresponded to a production rate of about 425 tons/h. Bit lives averaged about 5000 ft. Experiences with different types of hammer showed that the outer sleeve in certain makes has a disconcerting habit of cracking under the prevailing conditions at Tunstead. This has been attributed by one manufacturer to the use of a hydraulic motor for the rotary motion, instead of compressed air. In certain hammers a more probable explanation is that the hammer shock is transmitted through to the sleeve, instead of the rock, when a cavity is encountered. Experimental work still continues, but despite the high cost of hammer replacements the choice of drill has been fully justified.

BLASTING

Ammonium nitrate is used for primary blasting in all the I.C.I. quarries. Fig. 3 gives the layout of a shot fired at Tunstead quarry during the transition stage from vertical to sloping holes. The term 'Diamon' is the local expression used to denote ammonium nitrate–diesel oil mixtures.

Trouble is met from time to time in sections of the quarries where the bottom of the face is not completely fragmented by the primary blast. In some circles it has been claimed that this feature can be eliminated by the use of steeply angled holes, but it has not been possible to confirm this using holes inclined at up to 25° to the vertical.

The only advantage gained from firing off-vertical holes is that a cleaner face in certain sections of the quarry, i.e. a face with less precariously balanced pieces of rock, is left behind as a result of the scouring action of the blasted rock on the face. Above a certain angle this advantage can be offset by the tendency for small pieces of rock to roll down the slope and be projected some distance horizontally when they reach the quarry floor. Apart from this many quarries have closely spaced vertical and horizontal joints and the face resulting after blasting, while approximating to a straight line in section, actually consists of a series of steps. Above a certain limiting angle individual pieces of rock can be left balancing dangerously on these steps as the main rock heap slides down the quarry face. This is a potential danger which has had to be borne in mind in sections of the quarry when experimenting with holes more than 20° to the vertical.



WELL BLAST Nº 916 — SITE A.
10 HOLES 4¾" OR 5" DIAM. DRILLED AT 3° SLOPE

HOLE Nº	FACE HT FT	HOLE DEPTH FT	BURDEN (FLOOR LEVEL) FT	TONNAGE	RATIO TONS/LB.	TOTAL CHARGE LBS	BASE CHARGE			DECK CHARGES					TOTAL DIAMON	
							DIAMON	PRIMER		DIAMON						
							CWT	LBS PAGD		CWT	CWT	CWT	CWT	CWT	CWT	LBS
1	172	180	16	4550	5·07	898½	3	2½		2	1½	1	½		8	896
2	172	180	20	4700	4·65	1010½	4	2½		3	1	⅞	⅛		9	1008
3	172	180	20	4700	5 15	912½	3	2½		2	2	1	⅛		8⅛	910
4	172	180	20	4250	4 66	912½	3	2½		1½	1½	1½	½	⅛	8⅛	910
5	172	180	20	4450	4·88	912½	3	2½		2	2	1	⅛		8⅛	910
6	173	180	20	4700	5·15	912½	4	2½		2	1½	½	⅛		8⅛	910
7	173	180	21	4500	4·93	912½	3	2½		2	2	1	⅛		8⅛	910
8	173	180	21	4500	4·93	912½	4	2½		2	1½	½	⅛		8⅛	910
9	173	180	21	4700	5·15	912½	3	2½		2	2	1	⅛		8⅛	910
10	173	180	21	3000	4·36	688½	2	2½		1½	1	½	½	½/⅛	6⅛	686
		1800		44050	4·92	8985	32	25		20	16	8⅞	2⅜	⅝	80	8960

AVERAGE SPACING: 16'-0" (EXCLUDING CUT-OUT HOLE) | AVERAGE BURDEN: 20'-0" AT FLOOR LEVEL

INITIATION: DETONATING DELAY RELAY (15-MILLI SECS) BETWEEN EVERY OTHER HOLE

EXPLOSIVE PLANNED: { BASE DIAMON 32 CWTS. DECK DIAMON 48 CWTS.
HIGH EXPLOSIVE PRIMER: 25 LBS POLAR AMMON GELATINE DYNAMITE

DOUBLE CORDTEX DETONATING FUSE: 1750' | TONS/FT. DRILLED: 24·5

Fig. 3.—Typical design of blast.

Toes, or bottoms, when they occur, are dealt with at Tunstead by drilling 2½-in holes, a recent development being the design of a console which enables the drill to be hydraulically controlled from a point some 60 ft away. Conventional high explosives are being used, but permission is now being sought from the Home Office to use ammonium nitrate–fuel oil blown into the holes using compressed air.

At another of the smaller quarries it has been intended to quarry to within 300 ft of dwelling houses, and ground vibration has therefore received a lot of attention. For some considerable time horizontal vibration has been measured using a vibrograph developed within the Company. This has been done on each occasion a primary shot has been fired and a study of a large number of results has enabled vibration limits to be forecast with certainty in any section of the quarry for a given design of blast.

Another interesting line of investigation is in connection with the noise produced in blasting operations and at present noise volume is also being measured, together with an accurate observation of weather conditions.

LOADING

Mechanized loading actually started in 1918 in a small quarry adjacent to the existing Tunstead quarry, but operations ceased when it was found that the provision made for dealing with the dirt in the mechanized feed was inadequate. In 1945 partial mechanization was introduced at Tunstead, subsequent to the development of a scrubbing process which enabled the plant to deal with the mechanized all-in feed. In this scheme a 2½-yd^3 excavator loading into 10-ton end-tip dumpers, which fed a 72-in by 48-in jaw crusher, was used. Subsequently, 3½-yd^3, 5-yd^3, and ultimately 6-yd^3 shovels came into operation. The 5-yd^3 excavators are still in good operating condition, but their replacement was justified on increased loading rates and, probably more important, on the radical change in design which it was considered would reduce maintenance costs; this has proved to be the case in practice. The outdated excavators have been retained on site as they are of much more value as spares than the price which has been offered for them. Apart from this, at Tunstead in particular, long traverses of modern machines, necessitated by the occurrence of mineralized faults, are often eliminated by the strategic placing of old machines at selected loading sites.

Close contact is maintained with the excavator manufacturers and they are aware of sundry minor modifications which have been made to the original machines, some of which are now available either as standard equipment or as optional extras. These include additions or alterations to various parts of the lubricating system, which are operated in conjunction with a mobile lubricating unit built in the quarry. This has appreciably increased the time the excavators are available for work and has decreased the operating manpower in this section.

Loading rates can only be considered in relation to other factors, such as quarry haulage, rock fragmentation, etc. Detailed statistics are kept, but these frequently vary, not owing to excavator troubles but to temporary lack of quarry transport and local variations in digging conditions. Loading rates of the 6-yd^3 machines taken during actual loading operations in 1962 averaged 432 tons per running hour. Most operators, however, are more interested in the loading rate per hour normally clocked by the excavator and vehicle driver, which includes lost time for all reasons. In doing plant estimates a figure of 290 tons/h for 6-yd^3 machines is used—which has been proved in practice; this figure, of course, covers stoppages due to the crushing plant, lack of transport and other reasons.

The 6-yd^3 excavators have a rope crowd, necessitating rope renewals at approximately every 116 000 tons. The corresponding figure for the hoist ropes is 203 000 tons.

As auxiliary to the main loading operations two large modern rubber-tyred shovels are operated and act as 'maids of all work'. They were originally justified on their extreme mobility as pieces of bulldozing equipment and as replacements for much smaller and hard-to-maintain hydraulic shovels which were used to load out concrete aggregates. One of their main duties is pushing up edges of newly blasted heaps towards excavators which otherwise would be operating inefficiently. They are also used for the removal of overburden and are generally in very high demand.

TRANSPORT

Transport from the quarry face to the various crushing plants is by means of 50-ton side-tip vehicles at Tunstead, 15-ton end-tippers at Raynes and 20-ton end-tippers at Hindlow. In the Buxton area the vehicles have been designed as an integral part of the loading and crushing operation, having been related to size of excavator, crusher and quarry layout, which gives level hauls from the excavators to the crushing plant. Work-study techniques were also widely used in the process. At Tunstead, apart from the period covered by partial mechanization, 20-ton side-tippers, in conjunction with $3\frac{1}{2}$-yd^3 excavators, were used initially. The haulage unit, which was powered by a succession of different makes of engine, each developing about 90 hp at 1000 rev, was manufactured to the Company's specification, while the trailer part of the unit was made by various contractors.

Progress was made from 3½- to 5-yd³ excavators, which were soon manufactured in this country, and it became evident that larger-capacity vehicles would be advantageous. This was accomplished on an interim basis by extending the sides of the existing trailers to enable them to take 30 tons, but in due course thought was given to the maximum size of vehicle which could be geometrically accommodated by the 60-in gyratory crusher and would allow full advantage to be taken of even larger excavators. Eventually, for trial purposes, such a vehicle was made by the simple process of inserting an extra piece into the length of the 30-ton vehicles. It was found that the maximum tonnage which could be loaded into such a vehicle was 60 tons, although with size variation which occurred on different parts of the loading site it was appreciated that the average would be about 45 tons. The data obtained from this experimental vehicle enabled the Company to design with confidence a vehicle having these capacities for particular requirements.

It was found that a combination of I.C.I.'s own design, coupled with the fact that the trailer could be fabricated in the Company's own workshops, gave a vehicle which would fulfil requirements and which was substantially cheaper than those available from manufacturers. Careful method-study observations showed that with provision for a spare, and with three 6-yd³ shovels in operation, eight vehicles would be required, and this actually is the present fleet at Tunstead. With the eight vehicles and three 6-yd³ shovels there is a potential quarry input to the crushing plant of about 4 000 000 tons per annum, working a conventional 11-shift week.

One of the expensive items of such vehicles is tyre maintenance, which at present is running at about 0·25d. per ton. Experiments have been carried out on a large number of different tyre makes and treads, and the Company has its own vulcanizing team (who also attend to belt repairs) and large platens for the purpose. It has been found that careful attention to operating conditions, in and around the shovel in particular, more than pays for itself. The quarry floor is built up to near level before the excavator moves forward and a small rubber-tyred bulldozer moves about from one digging site to another to keep the area free of large stones.

A system of arterial concrete roads has been built in the Tunstead area. In the smaller quarries, where material for 'blinding' has always been readily available and there is only a relatively short haul, such roads do not appear to be necessary; indeed, in at least one of the smaller quarries the technique of making and maintaining roads from quarry material appears to have been mastered, without the disastrous potholing which used to occur during continuous heavy rainfall. The lesson has been applied to the largest quarry and, moreover, is one (but by no means the only) case where the flow of technical know-how from Tunstead to the smaller quarries has been reversed.

Throughout the years there has been a progressive decrease in transport costs, the most spectacular reduction being on maintenance. With so many variables changing over the period it is extremely difficult to apportion this improvement quantitively to different factors. The improved fragmentation which has taken place progressively through the years is unquestionably a major factor in the reduction, particularly of maintenance costs. Without question the policy of maintaining a level floor has also been a major item but, while these two factors have undeniably had a substantial effect, the improvement to the design of the vehicle has been the greatest single factor.

CRUSHING PLANT

The diagrammatic layout of Tunstead crushing plant is given in Fig. 4.

At Tunstead primary crushing is by a 60-in Traylor gyratory crusher which was commissioned in 1951. Under average operating conditions, it is fed at the rate of 1200 t/h, the limitation being not the crusher but bottlenecks elsewhere in the plant. There is no installed spare, although vital castings are in stock and a three months' emergency supply of stone is held which could be augmented by stepping up output from other quarries.

The crusher (Fig. 5) has given very little trouble, with one exception which is worth noting. In 1955 the 53-in diameter eccentric bearing

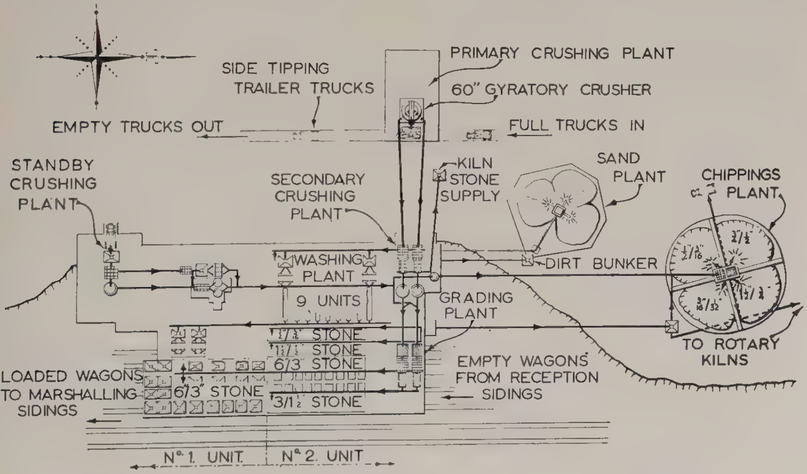


Fig. 4.—Simplified crushing and screening plant layout at Tunstead quarry.

Fig. 5.—Bowl of the 60-in gyratory at Tunstead crushing plant.

overheated after crushing 14 500 000 tons. At the time, so far as output was concerned, this gave rise to no great anxiety as a spare made and inspected in the U.S.A. was immediately available.

Until that time concern had only been felt about the gear meshing, and the bearing and gear assembly had been installed and rotated only sufficiently to enable this to be checked, but no trial had been carried out while the unit was actually crushing. On replacing the defective bearing and gear assembly with the spare it was found that while behaving satisfactorily when running light, the temperature of the pedestal bearing rocketed immediately stone was fed into the crusher bowl. A wide variety of theories was put forward to account for this phenomenon, involving re-designs to the simple pedestal bearing which had previously been completely satisfactory in service. Finally, the trouble was traced to the fact that the machining of the white metal eccentric bearing was slightly in error, being 0·020 in. out of square in its total height of 7 ft. Two spare bearings are now held—which have been re-metalled and machined in the quarry workshops.

The simple lesson has thus been learned that vital spares cannot be considered as spares until they have run for some time under actual operating conditions. It is perhaps easy to be misled into thinking that some relaxation in specification and inspection of such large pieces of equipment is possible merely because of their size and the rough treatment to which they are subjected during operation. One interesting feature of one of the re-designs of the pedestal bearing was the incorporation of springs which would allow some degree of self-alignment to take place, and such a bearing has been in operation for some considerable time.

Some trouble was experienced initially with the pitting of gears, but this has been largely overcome by erecting a mock set-up in the workshops which simulated operating conditions so far as gear alignment was concerned. This allowed final dressing of gears to take place before installation.

Scrubbing unit

Without question the most important technical achievement made by I.C.I. in limestone quarrying has been the successful design of an economic scrubbing unit (shown diagrammatically in Fig. 6).

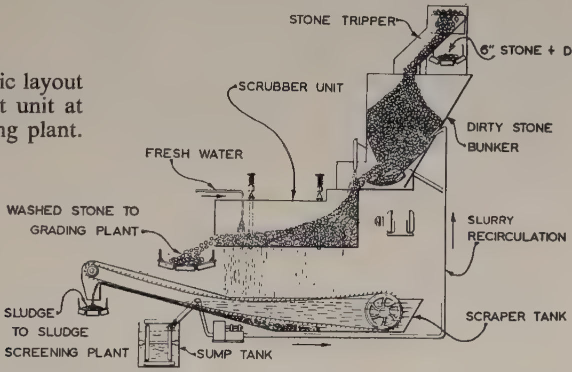


Fig. 6.—Diagrammatic layout of washing plant unit at Tunstead crushing plant.

Experience had shown that to produce chemically clean stone from a shovel-loading operation in I.C.I.'s particular quarries without throwing away considerable tonnages of the smaller fractions was impossible without adequate and economical washing.

After a long period of investigation a suitable unit was evolved. All quarry feed of —6-in ring size is passed through these scrubbers, the initial scrubbing being effected by recirculating slurry and the final rinse by clean water. The —$\frac{1}{4}$-in material, which passes through the perforations in the scrubber drums into the scraper tanks below, is then passed through a similar scrubber with drums having longitudinal apertures with gaps of approximately 0·03 in. This gives an extremely clean product for which there is plenty of demand. Finally, the 'throughs' from this unit are pumped to a settling tank and, in turn, the scrapings from the settling tank are pumped to a sand plant. The final 5 per cent of waste, which contains virtually all the clay from the quarry feed, is pumped to carefully maintained slurry ponds some distance away.

Associated with the crushing plant is also a gravels plant which produces a range of high-class concreting aggregates, of which large quantities are held in stock.

At Hindlow the problem of obtaining a high yield of clean stone from the quarry input has been complicated by lack of water and slurry disposal facilities, which have introduced serious difficulty in the way of installing scrubber units. This is overcome by sending $-2\frac{1}{2}$-in stone, which cannot be sold externally, to Tunstead and processing it there.

At Raynes quarry a scrubber unit identical to those at Tunstead is in operation, although the recirculation of the slurry as practised at Tunstead is not necessary as there are plentiful supplies of sea water. The final rinse, however, is with fresh water.

A further important mechanical feature of the crushing and screening operations in the Buxton Lime Works is the wide adoption of a two-bearing screen which obtains its vibratory motion from an out-of-balance shaft. This, of course, is not a new principle, but the Company was warned that trouble would arise in trying to adopt it for the large scalping screens, which have deck sizes of 8 ft by 16 ft. This has not been the case and the two-bearing screens are among the most reliable pieces of equipment on the works.

STOCKPILING

Stockpiling at Tunstead, except for kilnstone and other ancillary products, is in reinforced concrete bunkers which, in total, have a capacity of 20 000 tons of all sizes.

At Hindlow there was the problem of supplying a battery of seven

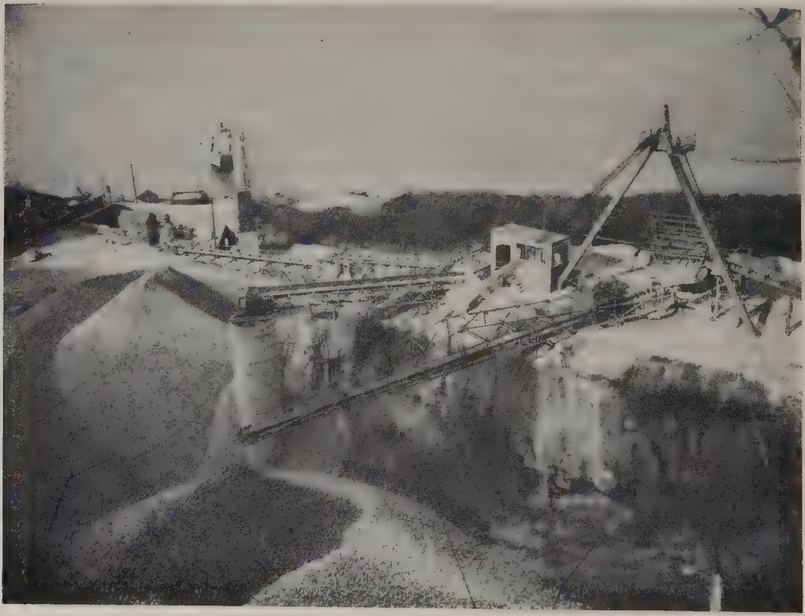


Fig. 7.—The stockpiling system at Raynes quarry.

high-capacity vertical kilns with clean stone of a suitable grading from a mechanized quarry. Experience with hand-filling had indicated what would be a workable grade and the plant was designed with this in view. In order to even out variations in grading in the kilnstone feed, however, due, in turn, to variations in the crusher feed, the stone is initially graded into three stockpiles containing different sizes. The kiln feed is then reclaimed at known rates from these piles by means of a remote-controlled feeder and conveyor system located in tunnels under the stockpiles and re-mixed to give the desired grading for the kilns.

A not dissimilar system is in use at Raynes quarry (Fig. 7), where the stone for ship-loading and rail-loading is again reclaimed by means of a conveyor system operating under the stockpiles. The stone for ship-loading is conveyed on to a jetty and transferred to a shuttle conveyor, which enables a ship to be loaded without any further manoeuvring once it has tied up to the jetty. A feature of the system is the terminal wing conveyor, which facilitates the spreading of the load across the width of the ship's hold. The end of the conveyor can also be readily located in a vertical plane to minimize breakage of stone which would otherwise occur.

SAFETY

Supervisory staff in the quarries are helped in their accident prevention activities by the works safety officer. Safe working, however, is clearly their responsibility and not that of the safety officer, and the managers are at liberty to accept or reject his advice at all times.

The first full-time safety officer for quarrying and ancillary activities was appointed in 1945, when the foundations of the present accident prevention techniques were laid. Since then the record has been one of continuous improvement. The frequency rate for lost-time accidents for individual years, computed according to British Standards, is given in Fig. 8. In 1962 this frequency rate reached a new low of 0·38 which, broadly speaking, means that if that performance is maintained only one man in three is likely to be off work once as a result of an accident throughout the whole of his working life. The production of successful accident prevention figures on this basis is, however, only of secondary importance in reducing accidents and, at best, is to be regarded as a method of keeping the score.

Before mechanization accidents directly associated with quarrying, generally at the quarry face itself, formed a major part of the total, but that is no longer the case and analysis of the accidents shows they are not of a type peculiar to quarries. Indeed, it is felt that a phase in accident prevention has now been reached when it can be said that there is no basic reason for regarding a quarry as a more unsafe place to work in than a factory. Further, while I.C.I. traditionally regards

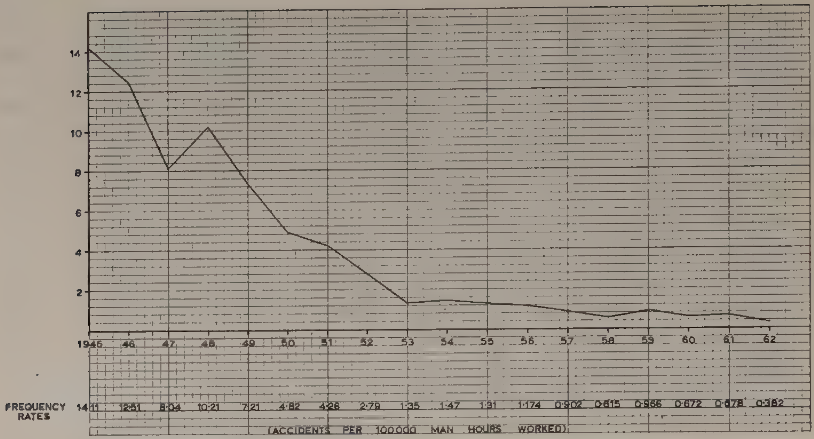


Fig. 8.—Accident frequency rates for quarries in the Buxton district and North Wales.

statutory guarding as a minimum requirement, it is doubtful whether the substantial sums spent make a major contribution to accident prevention. Indeed, it is often argued among the managers that activities in this respect could well militate against it, removing—as they do—some of the necessity for mental alertness.

Unquestionably, it is maintained that the most effective activities are those designed to improve the mental attitude of the individual worker to his job. Within the last two years, for example, in addition to other safety courses every member of the payroll has been taken from his job for one whole day to attend a course arranged to stimulate his safety consciousness. For some time the works have been analysing in detail all the jobs which have to be undertaken in the quarries, with the specific object of focusing attention on any safety hazards. These studies are known as 'job safety breakdowns' and are carried out by a combined committee of staff and payroll.

To quote one of the works councillors well known for his fervour on the subject of safety: 'We are primarily interested in preventing an accident before it happens'. A typical example is the rather unconventional lines of shunting locomotives (Fig. 9) which were modified as a result of concern at the poor visibility on many standard locomotives and the lack of a safe riding step for shunters. Both of these factors, in addition to human failing, played some part in two serious lost-time accidents which occurred within a period of years. In the locomotive prior to modification the head of a man 5 ft 4 in. tall standing in the middle of the track could only just be seen when he was 83 ft from the front buffers; nearer than that he was completely invisible to the driver. Subsequent to modification this distance was reduced to 10 ft.

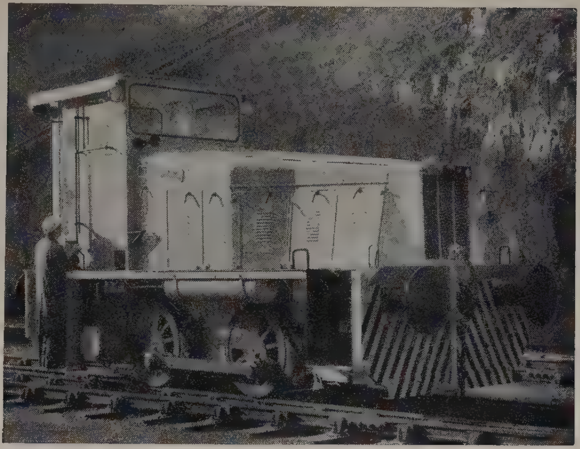


Fig. 9. — Modified standard gauge locomotive to give extra visibility.

When considering safety in quarrying practice the layman's mind automatically turns to the hazards involved in the handling and use of explosives. Over the last 20 years only one such accident has occurred, this being due to a blatant disregard of well publicized safety rules. It is always considered that the use of ammonium nitrate has reduced the possibility of this type of accident and, no doubt, positive refusal to use electric detonators down primary blast-holes is also another factor. In addition, the Company has compiled a rather voluminous code of practice in the handling and use of explosives, with special reference to its own quarries.

ORGANIZATION AND MANPOWER

Some reference has already been made early in the paper to the organization of I.C.I.'s quarrying and other ancillary activities. At the time of writing the works manager in charge of quarrying operations and associated activities is responsible to a Product Group Director of the I.C.I. Mond Division. Directly responsible to him are four senior managerial staff, all with high technical qualifications, one of whom is directly concerned with quarrying, the remainder being in charge of maintenance, limeburning, etc., and the workshops where much of the equipment is made.

The quarry manager and his assistants are trained mechanical engineers and, as far as possible, the policy is for the appointment of men trained in engineering for supervisory jobs down the line. As a generality, it has been found easier to teach time-served mechanics the principles of quarrying than to teach purely quarry-trained men the principles of engineering. Comprehensive basic training courses for foremen are available within the Company and from time to time short specialist courses are held.

It is difficult to split the total manpower of the works into individual operations as many services are common to quarrying and other extensive production activities. For example, there is no definite allocation of the maintenance team to quarrying alone as the same team maintains all plant and equipment on all the works.

Practically all the quarry payroll are on an incentives system. A Work Study Section of six staff operates as an integral part of management. They are not wholly occupied on the introduction of incentives. but probably spend at least half their time on method studies, in collaboration with the quarry management.

CONCLUSION

The improvement in efficiency in the I.C.I. quarries has been a progressive operation, particularly since the major breakthrough enabling the Company to clean at low cost large tonnages of the smaller sizes of stone which otherwise would have to have been discarded.

It is the object of every quarry manager to make profitable use of all the material removed from the quarry face and, measured in these terms, it is expected that the I.C.I. quarries in the Buxton area will soon be one hundred per cent efficient.

Productivity has also very substantially increased over the years and, simultaneously, the safety record has been greatly improved. Apart from major developments there is no doubt that the close and continuous attention to detail exercised by all workers will effect still further improvements in places where this seems at present to be almost impossible.

Acknowledgements.—The author is indebted to various members of I.C.I. staff for help and criticism during the preparation of this paper, and also to the Alkali Division (now part of Mond Division) of Imperial Chemical Industries, Ltd., for permission to submit it to the symposium.

Discussion

(Papers 10–12)

Mr. B. G. Fish,* introducing Papers 10 to 12, said that they were concerned with the techniques of hard rock quarrying. American, Czechoslovakian and British practices were described and that broad international coverage led to a number of interesting comparisons.

The common thread running through the three papers was the choice of blasting technique, although that was only one of the matters considered in the paper by Mr. Jackson. It was, however, in that particular field that similarities and contrasts could be especially noted, and around which useful discussion could perhaps be stimulated.

The paper by Professor Kochanowsky, while aimed particularly at blasting techniques, also quite properly took in drilling techniques—the two aspects being in many ways inseparable. The adoption of angled holes, for example, to achieve off-vertical faces had given rise to new designs of drilling machines. Also, the need to increase drilling productivity had led to the use of operating air pressures much in excess of the traditional maximum of 100 lb/in^2. That latter development, particularly applicable to down-the-hole hammers, was of considerable potential importance in the United Kingdom, where the tendency to use holes of about 4 in. in diameter had resulted in the very widespread use of such equipment.

Professor Kochanowsky included in his paper an interesting study of the interaction of the variables of bench height, bore-hole diameter, burden and spacing. A factor of importance in that respect was the scale of operations and thus the size of excavator buckets. Secondary breakage using explosives was expensive and the tendency in Europe was now to avoid that method and adopt the drop-ball, where possible, in order to minimize labour costs. That was one reason why relatively small holes at close burdens and spacing were preferred. The encroachment of urban development on some quarry operations, and the subsequent limitation on permissible ground vibration, was another reason for that choice.

The benefits of AN–FO and slurry blasting, in terms of cost, were clearly brought out. That aspect would no doubt provide material for discussion, but in order to avoid any misunderstanding it was important to recognize the legal aspect. In the United Kingdom they had been urged from many quarters to adopt slurry blasting. The fact was that in the United Kingdom the regulations did not allow simple on-site mixing of such materials when one of the constituents was itself an explosive. Thus, although the on-site mixing of AN–FO was permitted and well established—by virtue of the fact that the two constituents were not in themselves explosive materials—the use of slurry necessitated the construction of a 'factory' meeting quite strict regulations. It was only on the very largest of the operations that the establishment of such a factory began to be either feasible or economic.

---

*Institute of Quarrying, London.

The paper by Dr. Lukeš presented a considerable contrast. The author was concerned primarily with the rapid and economic development of new large-scale quarry operations, in the face of the lack of suitable and efficient drilling rigs and the necessity of avoiding high capital investment. In those circumstances the application of heading blasts was particularly appropriate and considerable study had gone into their design and use. Once the new quarries had been opened out by that method, it had been found appropriate to combine the use of heading blasts to deal with the top bench, which should be as high as possible, with multiple deep-hole blasts in the lower benches of moderate height.

A particular virtue of the technique was that a very rough quarry top did not affect the operation, since it was not necessary to level off the ground for the accommodation of drilling rigs.

A large part of the paper was devoted to a consideration of methods of calculating the charges for mass blasting. Beginning with the basic elementary relationship between weight of explosive and the volume of rock broken, the various refinements and modifications proposed by a number of authorities were considered. The empirical formulae became more and more complicated in the efforts to take account of all factors and to reach a relationship of universal application. The question remained as to whether local geological conditions, such as the occurrence of backs, slips and bedding planes, which might have an overwhelming effect in practice, could ever be reduced to a mathematical parameter.

Mr. Jackson's paper was a descriptive one, tracing the development of limestone quarrying practice by one company in the United Kingdom. Of special relevance to Session 3 of the symposium were the sections on drilling and blasting, and particularly interesting was the conclusion, reached after intensive investigations, that the hole size should be $4\frac{3}{4}$ in. in diameter. Of special significance was the fact that the face height involved was one approaching 180 ft.

The introduction of an inclined face coincided with the application of pure rotary drilling machines. The disadvantage of hole deviation, of special importance in deep-hole drilling, immediately made itself felt. That was presumably due to the thrusts required to maintain a reasonable penetration rate. A change to down-the-hole hammer drilling overcame that difficulty.

Blasting experience endorsed the use of inclined holes for primary work, but the sole advantage observed was that of maintaining a cleaner, and thus a safer, face. It was interesting to compare that view with the much wider claims made for inclined faces by Kochanowsky and others. In particular, the claim of better breakage at the toe had apparently not been confirmed. Satisfaction was expressed in the technique of combining down-the-hole drilling, at a relatively small hole diameter, with the use of on-site mixed AN–FO explosive.

The operational matters other than drilling and blasting referred to by Mr. Jackson were important and had a considerable bearing on the extractive techniques employed. Thus the development of a reliable means of cleaning the stone by scrubbing paved the way for the introduction of full mechanization, without which meticulous selection of material at the face was out of the question.

Again, the choice of a very large primary crusher had allowed the use of excavators fitted with 6-yd^3 buckets and minimized the necessity for secondary breakage at the face. The use of large excavators had, in turn, led to the development and application of quarry transport vehicles of 50-t capacity.

Finally, the safety experience described by Mr. Jackson should be of considerable value to other companies. That the accident prevention activities had been successful was strikingly demonstrated by the graph of accident frequency rates (Fig. 8, p. 276). In particular, the use of 'job safety breakdown' studies recommended itself for wider application in the industry.

Mr. D. L. Ward,* referring to Paper 10, said that it appeared that bottom priming initiation of slurry-type bottom charges in long, medium-size (4 in.) holes might well affect the resulting detonation owing to the high speed of detonation of the Cordtex passing alongside the slurry. He asked if the author would comment on that and perhaps give details of the methods of initiation in the U.S.A.

Mr. Jackson mentioned (p. 266) that his company decided not to use tractor-mounted drills because of track maintenance, due in turn to the rough quarry top. He thought that that maintenance cost was offset by the ease of setting up the drilling machine and by using its own power at all times and not having to use a further vehicle for moving the machines, as was the case with rubber-tyred rigs.

---

*Roads Reconstruction (Quarries), Ltd., Wiltshire.

He would welcome the author's comment on the method of initiating the deck charges as shown in the sketch of a typical hole charge (Fig. 3, p. 267). It did appear that the Cordtex initiated those charges, which must surely desensitize the AN–FO mixture. He wondered if the 'toe' holes drilled by the percussive machine were blasted with the main face holes.

Mr. A. Shadmon* said that he had been following the application of angle drilling for many years and although results seemed to vary, in general, the method showed certain advantages.

The relationship of blasting to the thickness of beds and to the direction of the dip were factors which were not considered in the papers by Professor Kochanowsky and Dr. Lukeš, although they did discuss bench thickness and the nature of plane partings.

The bed thickness was surely of importance since, given similar conditions, the consumption of explosives for blasting thin-bedded strata was quite different from the consumption in thick-bedded strata, not to mention variation in fragmentation. Again, given a virgin hill-site, and the choice of development directions, what would the authors suggest as being the more efficient—a working face along the strike or one along the dip?

Mr. T. T. Heywood† said that he would welcome Mr. Jackson's detailed comments on exactly how 'in at least one of the smaller quarries the technique of making and maintaining roads . . . appears to have been mastered' (p. 270).

Any organization having the facilities to design and build its own trucks was to be envied, and he would be grateful if Mr. Jackson could give more details of the truck I.C.I., Ltd., developed, together with his views on the advantages and disadvantages of articulated trucks compared with the conventional fixed-chassis type.

Mr. T. M. Dover‡ asked Professor Kochanowsky to comment on the theoretical effect of well developed vertical jointing and horizontal bedding planes on the drilling and blasting of rock in thin beds (10 to 30 ft vertical thickness) when using inclined blast-holes.

---

*Ministry of Development, Jerusalem.

†N. V. Mijn Maatschappij, Curaçao.

‡Ore Mining Branch of The United Steel Cos., Ltd., Lincolnshire.

Experiments in drilling Northampton sand ironstone with rotary holes $2\frac{3}{8}$ in. in diameter, inclined 22° to the vertical, gave higher bit wear when collaring holes, more difficulty in changing drill rods and an increase in the time required for charging and stemming shot holes than when they were vertical. He wondered if those disadvantages were normally found when using inclined holes.

Mr. G. G. Riley* mentioned that Mr. Jackson had indicated that deviation with rotary drilling on a high face could become alarmingly high. He wondered if the author could indicate the extent of deviation with down-the-hole drills in the same conditions.

Turning to Professor Kochanowsky's paper, AN–FO mixtures were used quite extensively, where conditions permitted, in British quarries. Over the period of their introduction during the past seven or eight years blasting ratios in terms of tons of rock per lb of explosive had, however, risen fairly sharply. With conventional explosives blasting ratios were commonly in the 4 to 5 t/lb range. With AN–FO, on faces 40 to 100 ft in height, heavier blasting ratios in the 2·5 to 3·5 t/lb range were normal and a trend to even higher ratios was evident. With the adoption of AN–FO, therefore, the quarry operator should not ignore the fact that his drilling costs per ton of production were liable to increase by virtue of smaller hole spacings and burdens.

With regard to slurries, that type of explosive was not particularly novel in respect of the energy-producing ingredients. Indeed, slurry explosives were based on classical powder explosive formulations incorporating TNT as a sensitizer, oxidizing agents, fuels and, in some cases, metallic fuels of high efficiency which boosted the energy developed. Dry formulations of that type had been in regular use for many years. The addition of 10 to 15 per cent of water to render the mixture viscous raised doubts as to the adequacy of the margin of sensitiveness in explosive terms. It was well appreciated that powder compositions were particularly prone to desensitization by water. Sensitiveness to initiation was one problem, but perhaps more important was sensitiveness to propagation throughout a column charge in a blast-hole. That was a function of blast-hole diameter and for smaller diameters it appeared that slurries would become costly because of the need for compositions containing high percentages of expensive sensitizers. Could the author indicate the minimum blast-hole diameter in which slurries could currently be used with an adequate margin of sensitiveness.

---

*Imperial Chemical Industries, Ltd., Renfrewshire.

The main advantage of slurries would appear to lie in their ability to fill the full cross-section of the blast-hole rather than in superior explosive properties. A rough calculation indicated that a 40 to 50 per cent increase in blast-hole loading density (lb/ft) should be attainable compared with AN–FO or cartridged explosive. Wider blast-hole spacings for holes of a given diameter, and thence lower drilling costs, were thus obviously possible.

Recent information from Canada suggested that slurry costs varied from 18–22 cents per lb, which was in line with more conventional fixed explosives. Professor Kochanowsky mentioned 7–18 cents per lb and that seemed rather low. He would be grateful for further comments from the author.

Mr. T. M. Templeton,* referring to Professor Kochanowsky's remarks on the advantages derived from angle-hole drilling, wished to point out that in some instances drilling efficiency might actually be impaired when too shallow an angle was employed. For example, when in a material of high specific gravity—such as hematite—evacuation of drill debris from the hole became less efficient when the angle was in excess of 20° to the vertical.

With angles greater than 20°, particularly in large-diameter holes, there was a tendency for the debris to accumulate on the footwall of the hole, thus diminishing the efficiency of the flushing medium. In his own experience that feature was common to wet and dry drilling and also to conventional percussion machines and down-the-hole hammers.

Presumably the current trend towards air pressures of around 200 lb/in^2 would do much to obviate that problem. He would suggest, however, that the opencast or quarry operator, when adopting inclined blast-hole drilling, should consider, in addition to the general advantages of the technique, the inclination most suitable for the material with regard to drill performance.

Professor Kochanowsky stated (p. 237) that with AN–FO explosive it was the shape and size of the prilled or flaked ammonium nitrate which determined its explosive strength. In Southern Africa a dense-prilled (non-porous) ammonium nitrate and a calcium–ammonium nitrate prill coated with diatomaceous earth were available to the mining industry. In addition, his Company imported a quantity of the Spencer

---

*Squirrell and Popplewell, Ayr, Scotland.

Chemical Company's 'N–IV' porous prilled ammonium nitrate from the U.S.A., and had demonstrated to their satisfaction that the Spencer porous prill was clearly the most effective from the point of view of strength.

To him that indicated that, other factors being equal, i.e. confinement, percentage of fuel oil, etc., it was the intimacy of mix which was the primary factor in determining the strength of AN–FO mixtures; with AN–FO explosives the intimacy of the mix was a function of porosity.

Dr. B. O. Skipp* said that while Dr. Lukeš referred in his paper to various rock strength parameters, he did not mention tensile strength—a factor he would have thought relevant. The measurement of a meaningful 'rock strength' was difficult and he was not surprised that that factor did not seem to have received much attention in Western practice. In contrast to the practice described by Dr. Lukeš, it seemed that local experience rather than generalized formulae was used for blasting calculations in the West.

Mr. M. W. Rushton† said that he would like Mr. Jackson to elaborate on the description of the scrubbing unit described in his paper (p. 273), particularly with regard to capacity, for instance the ratio for foreign material to clean stone recovered. Apart from material contained in the clay filled, what other sources of contamination occurred?

Finally, he would be grateful if some indication of the nature and amount of overburden to be contended with, and how that problem was tackled, could be given.

Mr. H. B. S. Dales‡ asked Mr. Jackson for further details of how chemically clean limestone was obtained in the quarry. It appeared that there were three impurities that had to be removed: (*a*) the clay resulting from the weathering of volcanic material parallel to the bedding planes; (*b*) the plastic clay filling the vertical joints; and (*c*) the barytes and galena occurring in the mineral veins.

Did the scrubbing unit deal equally effectively with both the volcanic clay and the plastic clay from the joints, and how were the barytes and galena removed? He wondered if they were screened out at the crushing plant or if differential quarrying were adopted in areas of the quarry where mineral veins were frequent or, indeed, if any other method were used.

---

*Rock Mechanics, Ltd., London.

†Nchanga Consolidated Copper Mines, Ltd., Chingola, Zambia.

‡Position unknown.

Mr. R. W. N. Dron,* referring to Professor Kochanowsky's comments on the use of high-pressure air, i.e. in excess of 100 lb/in^2, to operate down-the-hole hammers in the U.S.A., said that a machine of that type operating at 150 lb/in^2 had recently been demonstrated in the United Kingdom. While its performance was interesting, improvement on 90–100 lb/in^2 hammers scarcely warranted the increased capital cost involved. At the present time no reliable data on operating costs were available.

It would be of great interest to the quarrying and opencast industries in the United Kingdom to learn of American experience with machines operating at pressures of the order of 250 lb/in^2. He wondered if they were correct in assessing a 150 lb/in^2 unit as a very temporary product pending the satisfactory manufacture of units capable of economic operations at pressures appreciably in excess of 200 lb/in^2.

Reference had been made to the desire to adopt slurry explosives in place of AN–FO mixtures. While the basic composition of a slurry comprised only three ingredients—TNT or nitroglycerine, ammonium nitrate and water—the actual composition of a product acceptable to H.M. Inspectors of Explosives required at least six ingredients, and even up to fifteen different materials were included in the preparation of an acceptable commercial slurry. Since that led to a product costing substantially the same as conventional explosives, he wondered what advantages slurries held. It would appear that there was little real incentive to the manufacturer to develop slurry-type explosives for the hole diameter commonly used. Further comments on that view would be of great interest.

Mr. R. F. Lethbridge† said that the papers underlined the rapid developments in drilling and blasting techniques. Down-the-hole hammers had changed the whole pattern of drilling practice and the prospect of higher air pressures, resulting in even better performance,

---

*Dron-Dickson, Ltd., Glasgow, Scotland.
†Rio Tinto-Zinc Corporation, Ltd., London.

was also in view. The increased use of inclined faces in hard rock had already shown benefits both in economy and safety; moreover, the story was far from complete, and no doubt many developments in the field would come about.

The revolution in blasting practice, involving the adoption of ammonium nitrate blasting agents, was seen to be world-wide. Increased simplicity of operation and economy had resulted and, there again, developments were extending almost daily.

All that reflected a new approach by managements of quarrying and opencast mining operations, namely a willingness to experiment with new techniques and equipment on their own initiative and a refusal to be hamstrung by traditional practice.

Mr. J. G. Speake* said that Professor Kochanowsky had pointed out that in the United Kingdom hole diameters (and thus burdens) were generally smaller, and bench heights greater, than in the U.S.A. That meant, of course, that there was less margin for error in alignment of holes when changing from a nominally vertical face to an inclined face. Thus careful surveying and checking of hole location and inclination were essential in order to avoid the possibility of dangerously reducing burden.

The list of benefits claimed for inclined drilling was impressive and it seemed likely that most operators could take advantage of at least one of those. However, the claim that explosive consumption could be reduced by one per cent for every degree off-vertical was surprising, and one wondered how much evidence there was to support that. Elsewhere, a possible saving of about 10 per cent for a 20° inclination had been suggested and that seemed more realistic.

Professor Kochanowsky remarked (p. 229) that: 'It is generally assumed that the rock is broken by a shock wave reflected from a free face'. He would like to know if the author shared that assumption; there seemed to be some evidence that much of the overall blasting result was due to the reaction of the explosive gases on the burden.

WRITTEN CONTRIBUTIONS

Mr. S. Mahadkar:† The paper by Professor Kochanowsky gives very valuable information about advances in blasting techniques. I shall be grateful if the author will deal with the following points.

---

*Imperial Chemical Industries, Ltd., Warwickshire.
†Tata Chemicals, Ltd., India.

(*a*) No mention is made of the maximum (optimum) hole inclination which will give the maximum blasting efficiency. Can the author give this figure?

(*b*) It is reported that in the U.S.S.R. good progress is made in opencast blasting techniques—and a very good fragmentation of rock obtained—in blasting by employing a technique called 'air-decking'. This technique consists of separating column charges in various decks by 'wooden disks-cum-separator wooden rods' to allow air to act as intermediate stemming. During blasting the air acts to give a cushioning effect and the rock is well fragmented. This is because the whole of the air-deck utilizes the blasting energy derived from explosives' expansion, etc. I would welcome information on any such experiments carried out in the U.S.A.

(*c*) Professor Kochanowsky describes (p. 237) the composition of one of the slurries, but mentions that there are a number of other compositions. Details of other types of slurries would be of interest.

(*d*) It is claimed by experts that if a deep hole is charged with a high explosive column, e.g. gelignite, the blast would be better if the primer cartridge is located on top of the charge. If the primer cartridge is located at the bottom of the charge the blast is safe. Reasons given for these better results include the fact that the velocity of detonation at the contact point of Cordtex and gelignite is almost half that of the normal velocity of gelignite. It reaches normal after travelling for about 6 in. beyond the point of contact. Since in holes primed at bottom the Cordtex comes in contact with the entire gelignite column, the velocity of detonation throughout is half—giving not fully satisfactory results. Perhaps the author can suggest how the best results, with maximum safety, can be obtained in these circumstances.

(*e*) Can the author give any more information on high-pressure compressed air—a factor which may prove very useful for speedy work in quarries in the future?

Turning to Paper 12 by Mr. Jackson, the author mentions (p. 268) that it is intended to quarry to within 300 ft of dwelling houses. I would welcome information as to how ground vibrations are to be reduced and house-dwellers are to be protected from the incidence of flying rocks resulting from blasts. Can the author also give more information on the subject of the noise produced in blasting operations?

Do the 6-yd^3 excavator shovels have modified boom designs?

Finally, the author reports that there were no incidents of primary shots being misfired during the last ten years. In cases where the whole line of rocks is misfired and the Cordtex is found burnt, would relieving holes be blasted, or is there any other method of dealing with such a situation?

Mr. R. O. Morris:* Would Mr. Jackson say what equipment was used in his rotary drilling operations which led to such large deviations as 12 ft in holes only 180 ft deep? It is most important that the correct equipment, and especially the correct relationship between hole size and drill-rod size and rigidity, be used in rotary drilling. The equipment must also be related to the thrust available from the drilling machine and the experience and ability of the driller. When using a correctly balanced set of equipment rotary bore-holes of 180 ft in depth should be no more than 3 to 4 ft out of true.

Mr. J. G. Lancaster:† I shall be grateful if Mr. Jackson will give the size of the rubber-tyred shovels referred to in his paper, together with their capacities and any performance and cost figures available—the performances being related to the load of 15-ton end-tippers. I would also be grateful if the author could provide any tyre costs per ton of material moved. His experiences relating to the behaviour of such machines on sloping, wet, smooth ground when attempting to dig into solid piles would also be of interest.

---

*L. M. Van Moppes and Sons (Diamond Tools), Ltd.
†The Fullers' Earth Union, Ltd., Redhill, Surrey.

Authors' Replies

Papers (10–12)

Professor B. J. Kochanowsky: In answer to Mr. B. G. Fish's comment (p. 281) that 'the claim of better breakage at the toe had apparently not been confirmed', I would point out that it has, in fact, been confirmed by the National Gypsum Company, the National Lead Company, Canadian Johns-Mansville Asbestos Company and several others.

In reply to Mr. A. Shadmon (p. 282), it is not only the thickness of bed but also the mechanical property of rock which is decisive for the selection of an adequate blasting technique. Appropriate bore-hole diameter, type of explosive, burden and spacing are the first to be considered and this topic has been covered in my present paper.

Considering cases (1), (2), (3) and (4) in Fig. A, (1) and (2) are the most favourable. In case (3), after the blast the strata tend to slide down, causing the back break and endangering the drilling and loading operations at the bottom of the face. In case (4) the bench stands better but, especially when coyote blasts are used, it is difficult to adjust the explosive charge with the rock resistance, as well as to control the throw due to the inclination of the strata.

Mr. G. G. Riley (p. 284) enquires about the minimum blast-hole diameter suitable for slurry use. It depends upon the type of slurry; it is not difficult to have slurries up to 2 in. in diameter. Nitro starch slurries with very strong buster can go up to 1 in. in diameter. The price of slurry depends upon its composition. The price range of 7 to 18 cents per lb will cover most types of slurry explosives, where lowest prices are for weakest explosives over 4 in. in diameter. The price of metallic slurries and for cartridges smaller than 4 in. in diameter may go up to 23 cents per lb. In considering Canadian prices it must be borne in mind that the U.S. dollar has about a 10 per cent higher value than the Canadian dollar. The prices quoted in the paper were in U.S. dollars (cents).

Mr. R. W. N. Dron (p. 286) will be interested to learn that down-the-hole drills for 5- to $7\frac{1}{2}$-in bit diameter working with 250 lb/in^2 pressure are now available. It is too early to say anything definite about possible future use in quarrying and opencast industries of these high-pressure drills (operating at 150 lb/in^2 or over).

His reference (p. 286) to H. M. Inspectors of Explosives' requirements of up to fifteen ingredients for an acceptable slurry explosive seems very strange. I doubt very much if slurries used in other countries in Europe or America consist of so many ingredients.

In reply to Mr. J. G. Speake (p. 287), in another paper,* with reference to the cost of explosive and/or drilling, I stated: 'It appears that the saving is about one percent per degree of inclination'.

This statement has been confirmed in many cases in the quarries where inclined drilling method has been introduced. The names of these companies are listed collectively elsewhere.† The most impressive improvement in the U.S.A. has been achieved by the National Lead Company, the National Gypsum Company, the Tyrone Rock Products Company. Speaking of other countries, I know that Canadian Johns-Mansville Asbestos and Rheinische-Westfalische Company in Germany have achieved impressive results by using angle drilling. In most cases considerable cost reduction has also been observed in loading and crushing operations.

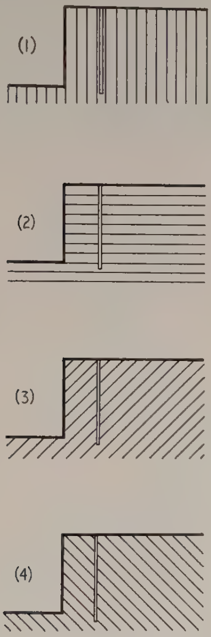


Fig. A.—Various rock strata inclinations.

---

*Kochanowsky, B. J., Theory and practice of inclined drilling for surface mining. *Miner. Ind., St. Coll. Pa*, **30**, no. 3, Dec. 1960, 5.

†See reference 1 on page 238.

The property of explosive, the powder factor, property, size, shape and constriction of the rock, the explosive loading density, and coupling ratio between the charge diameter and the bore-hole diameter, as well as several other factors, will determine whether the reflected shock wave or pressure caused by explosive gas expansion, or both, are responsible for rock failure. Since the constriction factor at the toe of the bench is the greatest, 2·5 to 4 times more explosive is required in the bottom charge compared to column charge to overcome the rock resistance at the bottom of the bench. It is therefore possible that in certain cases the reflected shock wave fractures the rock located at the bottom of the bench face.

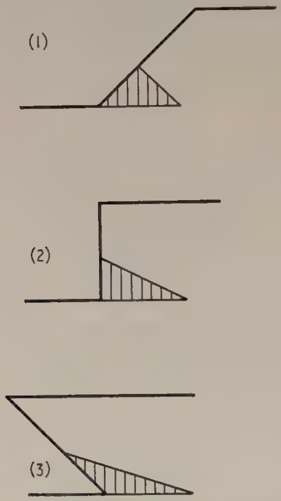


Fig. B.—Rock resistance against blasting versus bench inclination.

Also, however, when the primary cause of breaking rock located at the bottom of the bench is the pressure generated by explosive gas expansion, less force will be required because the rock resistance against blasting decreases with the increased bench inclination. As Fig. B shows, it is easier to break rock at point (1) than at (2) or at point (3). The rock in point (1) would have the least resistance and that at point (3) the greatest resistance against blasting.

In answer to Mr. S. Mahadkar (pp. 287–8), the greater the bore-hole inclination, the greater are the advantages which can be obtained with angle drilling. The following factors can limit this angle:

(1) Drilling of a bore-hole over 30° becomes more difficult.

(2) The loading of explosives packed in cartridges becomes more cumbersome. These difficulties usually begin with inclinations over 20°. This limitation does not apply when AN–FO explosives are blown by compressed air or slurries are pumped into the bore-holes.

(3) In some particular rock structure the inclined bore-hole may not stand as well as a vertical hole and may cause difficulties when explosive cartridges are introduced into the bore-hole.

(4) Where bench inclination is greater than 30° and rock contains clay or other sticky material, the rest of the rock pile [(1) in Fig. C] may remain on the slope of the bench and slide suddenly down creating a danger to the loading shovel (Fig. C). When the shovel is much bigger than the bench, this danger is not significant.

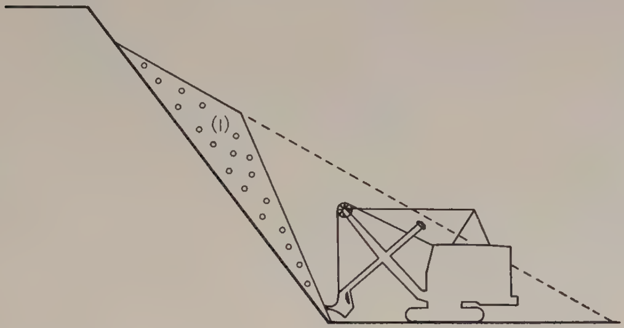


Fig. C.—Diagram showing potential danger due to sudden slides of rock when bench inclination is greater than 30°.

Inclination of 45° will give the maximum blasting efficiency. Because of the difficulties previously mentioned, 45° angle holes are seldom used in surface mining, but more often in road construction. The most practical bore-hole inclination in the mining and rock industry will vary between 20° and 30°.

In 1961 Professor Melnikov presented a paper* in which 'air-decking' was described and its favourable influence upon fragmentation was stressed.

Following this paper, we at the Pennsylvania State University initiated blast tests on concrete blocks where the influence of air gap in bore-hole axial direction has been investigated. Although these tests

---

*Melnikov, N. V. Influence of explosive charge design on results of blasting. In Clark, G. B., ed., *International symposium on mining research, vol. 1* (Oxford, etc.: Pergamon, 1962), 147–55.

are not yet complete, they seem to confirm the results obtained by Professor Melnikov. Presumably, this method is being used in actual mining operations, although no mention of it has been made in the pertinent literature. Whereas, prior to Melnikov's statement, air gaps had been used for many years in order to dilute the explosive power in bore-holes, these same air gaps may now be used to improve fragmentation in blasting.

Several types of slurries are manufactured in the U.S.A. The main ingredients are TNT, ammonium nitrate, water and guar gum. Instead of TNT, which forms about 20 to 30 per cent explosive, smokeless powder or nitro starch or composition B or in NCN slurry the powder of aluminium are used. Part of the ammonium nitrate can be replaced by sodium nitrate to make slurry more fluid and applicable in low temperatures. Gum is used to keep the slurry in gelatin form and the ingredient well distributed throughout the explosive. By varying the percentage of ingredients slurries of different strength can be obtained. The strongest slurries contain metallic component.

The best firing method with maximum safety would be the use of one primer at the bottom and another at the top and primecord along the column of explosive. In the case of AN–FO explosive located along a long bore-hole of small diameter it is preferable to place, a certain distance apart, one or more primers or high explosives which can be detonated by primecord.

In answer to Mr. Mahadkar's final question, this has been dealt with in my reply to Mr. Dron.

Dr. J. Lukeš: I was very sorry not to be able to attend the symposium. Clearly, parts of my paper have presented difficulties to the English reader—there being no exact translation for certain specialized Czech terms.

The difficulty of achieving a precise translation has perhaps been highlighted by Dr. B. O. Skipp (p. 285). On page 252 *p* is defined as the strength of the rock in tons/cm². The true explanation of this coefficient is the magnitude of tensile stress at the limit of plasticity (usually given in kg/cm²). I hope that this goes some way towards dealing with Dr. Skipp's remarks.

In reply to Mr. A. Shadmon (p. 282), it can be stated that the direction of development of a quarry face depends on the evaluation of many factors; hence every rock deposit requires as complete an evaluation as possible of all the complex factors involved. Results obtained even in the same deposit can differ considerably, such results depending, for example, on the particular blasting method selected. Where heading or chamber blasts are used, results will be improved if the charge is so placed as to act transversely to the direction of the strata. When deep-hole blasts are employed, however, drilling conditions may prove of overriding importance.

Mr. R. G. Jackson: In answer to Mr. D. L. Ward's query (p. 282) regarding the desensitizing effect of Cordtex on AN–FO mixtures, it is widely known that a 2 per cent mixture is more sensitive to detonation than a 5 per cent mixture, but the actual blasting effect of the former is less than the latter. Under certain quarrying conditions it is possible to make use of this property and eliminate explosive charges in the deck charges. Its success or failure depends, of course, on a number of factors, e.g. strength of rock to be blasted. The choice between rubber-tyred or track-mounted drilling rigs depends mainly on local quarry top conditions; the rubber-tyred drills chosen by the author's Company were, in fact, capable of travelling themselves.

With regard to the maintenance of quarry roads, referred to by Mr. T. T. Heywood (p. 282), these roads are made simply by laying down a base course foundation of coarse rock and binding with $-\frac{3}{16}$-in + 0-in material in a damp condition, and then running on the road. It is also important that the vehicles should not run on a set course for an indefinite time and that the road formation should be well drained. The 50-ton quarry vehicles developed by the author's Company are articulated, using a tractor built up by a commercial vehicle manufacturer from standard components. This tractor uses the largest available commercial bogie (British), together with an engine and torque converter transmission system to give the required performance on the level hauls in the quarry. The design of the trailer unit was based largely on experience in operating the range of smaller trailers already referred to, and comprises a rigid chassis carried on a two-axle bogie and having a robust all-welded body. A particular point was made to provide sufficient tyres to keep tyre loadings to reasonable levels, and experience has shown that this was fully justified.

The use of an articulated design was a logical development from previous experience, coupled with the fact that it enabled a 50-ton vehicle to be developed largely from existing components, as indicated above, at a low capital cost; moreover, articulated vehicles offer advantages of interchange of units for maintenance purposes.

The maximum extent of deviation from down-the-hole drills, under the same conditions, queried by Mr. G. G. Riley (p. 283), is approximately 7 ft.

In reply to Mr. M. W. Rushton (p. 285), the scrubbing unit consists of two horizontal perforated tubes which are vibrated by means of an out-of-balance shaft. This imparts to the load within the tube what can be best described as a 'boiling' action. Initially, fresh water is fed into the tube and the resulting slurry passes through openings into a slurry tank. The slurry thus formed is then used for an initial rinse and fresh water is only used for the final rinse.

In answer to Mr. H. B. S. Dales (pp. 285–6), the chemically clean limestone is obtained primarily by use of the scrubbers referred to in the previous paragraph. Impurities other than clay, e.g. galena, are dealt with by mixing off from various parts of the face and carefully sampling the small stone into which they tend to become concentrated. This presents no difficulty. The scrubbing unit deals equally effectively with both volcanic and plastic clay.

Dealing with the points raised by Mr. S. Mahadkar (pp. 288–9), a study was made of ground vibrations when quarrying a comparatively long distance away from the houses and formulae were developed for different parts of the quarry which gave the spacing and the charges permissible in blasting operations without damaging dwelling houses. There is no evidence that the blasts cannot be sufficiently well controlled to remove any possibility of house-dwellers being affected by flying rocks. Regarding noise, precautions are taken in the covering of the Cordtex on the surface with a heap of sand.

Regarding 6-yd^3 excavators, these shovels have modified boom designs. As indicated in the paper, the introduction of these excavators was based entirely on savings on maintenance, which have since been justified.

So far as misfiring is concerned, I know of no method other than that actually suggested; a series of relieving holes would have to be drilled and blasted.

Mr. R. O. Morris indicates (p. 289) that deviations of less than 4 ft in 180 ft should be attainable with correctly balanced rotary drilling equipment; this may be possible with the relatively expensive equipment (in terms of both capital and operating cost) which is used for oil-well drilling and prospecting. The rotary drills to which I referred were, however, designed for low-cost quarry operation and at that time were in common use in Continental quarries. The drill bits were multistage, tungsten carbide tipped wing cutters, giving $3\frac{3}{4}$-in diameter holes and the drill rods were $2\frac{3}{8}$ in. in diameter. One of these drills is still in use at one of the Company's quarries drilling 4-in diameter holes on a 100-ft high face with acceptable deviation.

The rubber-tyred shovels referred to by Mr. J. G. Lancaster (p. 289) have $4\frac{1}{2}$- and $2\frac{3}{4}$-yd^3 buckets. The larger machine is used for the heavier duties, such as overburden stripping, and the smaller machine primarily for loading concrete aggregates; for this work it has a loading rate of about 280 tons per hour into lorries of 12- to 15-ton capacity. No cost figures are available.

GROUP II—QUARRYING AND OPENCAST MINING OF MASSIVE DEPOSITS

Session 4—Transport and Equipment (Papers 13–15)

Chairman: Mr. F. Frith *Rapporteur:* Mr. T. M. Dover

622.619.4:622.271.3
629.114.4:622.271.3

Paper 13

Loading and Haulage in Quarries

PAUL FLACHSENBERG, Dipl. Ing., Dr. Ing. E.h.
Managing Director, Rheinische Kalksteinwerke und Dolomitwerke,
Wülfrath, Germany

SYNOPSIS

The considerations in the paper apply to compact, close-textured deposits which can yield outputs of 8000–12 000 metric tons of clean stone per day.

Of the external factors affecting loading and haulage, the crusher is the most significant; it determines the maximum capacity of the face-shovel bucket and of the haulage vehicles, as well as bucket dimensions (classifying action). Work with quarry faces 15–20 m high in conjunction with large-diameter bore-hole blasting makes for greater safety in loading and produces a rock pile that is very suitable for handling by the shovel.

Loading and haulage equipment affect one another and should be appropriately matched. Typical questions are: how large a capacity should be chosen for the shovel bucket and for the haulage vehicle in order to achieve a predetermined output, and how many vehicles are required, having regard to the factors governing a haulage cycle? From the numerous shovel/vehicle combinations, technically possible and suitably matched, the most favourable combination, in terms of economy, is selected with the aid of a comparative cost calculation.

The face shovel is the obvious loading machine for quarry use. The bucket is the most important component and governs the output of the shovel. The use of electrically powered drives makes smooth and uninterrupted operation possible.

Tractor loaders perform an essential auxiliary function in loading operations.

'Rail-less' haulage by means of heavy motor trucks embodying special design features has now become standard practice. A limiting value of 1 in 10 should not be exceeded for uphill and downhill gradients.

Human safety should take precedence over technical considerations of output and performance.

EXTERNAL FACTORS AFFECTING LOADING AND HAULAGE

The operation of a quarry is dependent on the deposit that is being worked. The extent of the deposit and the nature of the rock affect the layout and thus also constitute criteria for the set-up of the technical operations involved in loading and haulage.

The following considerations apply to compact, close-textured deposits which can be quarried at faces 15–20 m high, yield outputs of 8000–12 000 tons of clean stone per day, and attain an operating life of at least 30 to 40 years.

Assuming 20 per cent impurities, this output of clean stone corresponds to between 10 000 and 15 000 tons of loosened rock pile, so that with the usual 16 working hours per day and with allowance for time lost owing to breakdowns, handling capacities of 800–1200 tons of rock pile material per hour will be necessary. Fig. 1 illustrates the type of material and the method of working a deposit of this kind.

Loading the rock is preceded by drilling and blasting operations, and haulage is followed by crushing. The present paper is concerned only with the loading and haulage of loosened material (rock pile) suitable for further processing (ready for feeding to the crusher). It comprises the working of included foreign matter only as a fundamental feature and it does not deal with overburden removal.

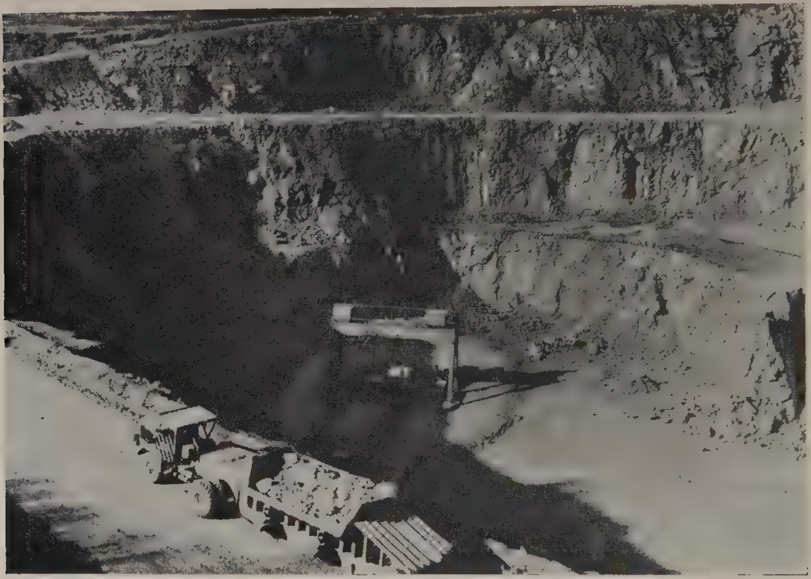


Fig. 1.—Compact close-textured limestone deposit.

The tectonic character, stratigraphical configuration and rock strengths are natural features which affect only a little the advantageous use of any particular type of plant for the operations to be examined.

On the other hand, foreign inclusions are important as they are often of such a kind, or occur in such quantities, that they cannot be separated by the screening plant. In that case it is not possible to obtain uniform crusher operation either, and for this reason the impurities are removed in special vehicles and are conveyed, separately from the stream of utilizable material, straight to the waste tip, i.e. by-passing the crusher ('selective quarrying'). Unfortunately, this procedure inevitably involves some loss of material.

Another natural factor that must be mentioned is the weather. Snow and frost are liable to disrupt haulage operation.

The requisite capacity of loading and haulage equipment should be determined, not primarily with reference to the actual winning of the material, but first and foremost with reference to the crusher, the capacity of which should be appropriate to the quantity of loosened rock to be processed. This capacity is in turn reflected in the loading and haulage equipment selected and in the drilling and blasting method employed. It affects the design features of the loading and haulage equipment, determines the maximum capacities of the face-shovel buckets and of the vehicles, and therefore also affects the number of face shovels and vehicles required. The number of loading points at the face is thus also governed by the crusher capacity.

The shovel bucket, or dipper, should be so dimensioned that it can handle pieces of rock only up to the largest size that can conveniently be passed through the crusher.

For a gyratory crusher with a feed opening of 1350 mm and a top effective shell diameter of 3500 mm, a lump of stone with edge dimensions of 900 mm should be regarded as the maximum feed size. The bucket shown in Fig. 2, which holds about 3 m^3, fulfils this requirement

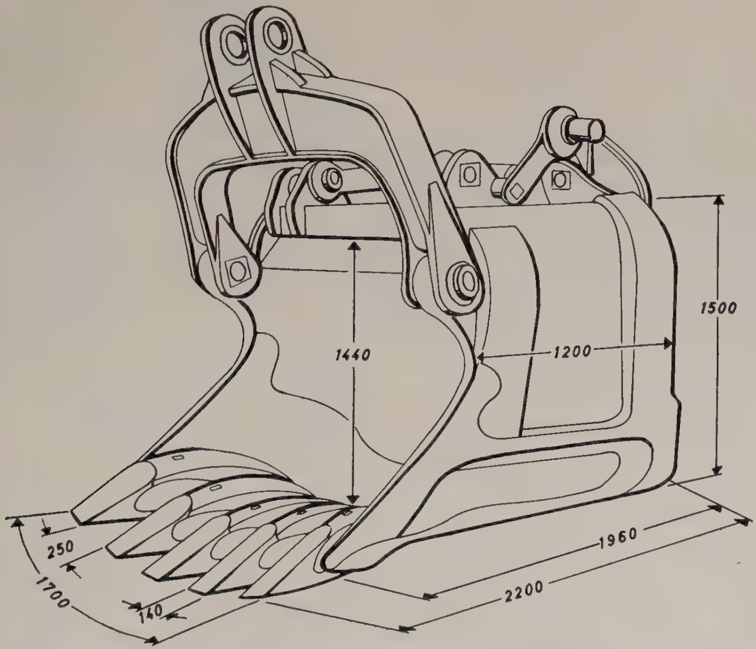


Fig. 2.—Face-shovel bucket of approximately 3·5 m^3 capacity (filled level).

dictated by the crusher and is practically unable to accept any larger pieces of material. The bucket size, in turn, gives indications as to the dimensional design of the other main components of the face shovel.

The question of providing adequate reserve capacity may also affect the decision on size of shovel. In this connection it must be pointed out that the size of the loading equipment should be limited in such a manner that the need for providing reserve capacity (as a standby to cope with variations in quarry output and with plant breakdowns) will not entail unnecessarily heavy burdens in capital costs.

The crusher also determines the maximum load-carrying capacity of the haulage unit. It is essential that the total vehicle load can be fed smoothly and briskly into the crusher. For example, if a crusher of the size already mentioned is used the capacity of the haulage truck should not exceed about 25 m^3, corresponding to a weight of 35–40 tons of rock. With the size of the vehicle thus determined, the nature of the haulage route—whether uphill or downhill or horizontal—will provide an indication as to the type of vehicle to employ.

In addition, some further comments may serve to show the importance of the crusher as the central governing feature: its location is, for one thing, determined by the existing or planned installations for the further processing of the quarried material—which, as a rule, are provided with direct road and rail access—and, moreover, by the most economical arrangement possible for bringing the crusher feed material from the quarry. The most economical method of conveying large quantities of material is by belt conveyor, but it can handle only pre-crushed material. For this reason the crusher is usually installed at the edge of the quarry and should, more particularly, be so located that, on the one hand, the rock taken from the quarry face can be brought to the crusher by the shortest possible route and that, on the other hand, the crushed material can be conveyed by a set of belt conveyors to the processing plant by the shortest possible route. It is assumed that the processing plant is also able to cope with varying crusher throughputs, which means that, generally speaking, it will not be necessary to provide storage facilities between the crusher and the processing plant. Ideally, the crusher (which is transportable) is installed at the rock pile itself and has the material fed directly into it by the shovel, so that the belt conveyor system is the only means of transport required.

Crusher performance can be improved by scalping off the fines before the material enters the crusher. This screening operation also helps to provide a more continuous feed of material to the crusher and thus ensures more uniform use. A jaw crusher must in any case be equipped with a feed device which must function as a surge receiver, but which need not also perform a screening function. In the case of a gyratory crusher—more particularly when dealing with loamy and cohesive impurities—a feed device is advantageous in that it makes for smooth and uninterrupted discharging of the haulage vehicles.

The dominating position that the crusher occupies in relation to loading, haulage and winning of the quarried material also remains unchanged when considered from the viewpoint of cost, since none of the operational procedures stands out prominently above the others in a comparison of the cost-producing factors under these four cost items. In the case of a quarry given as an example the operating costs (inclusive of capital charges) in respect of the items 'winning' (production), 'loading', 'haulage' and 'crushing' work out at 0.42, 0.36, 0.39 and 0.33 DM per ton respectively, so that the cost ratio is 28:24:26:22 per cent. This relationship is represented in Fig. 3. The cost of crushing —i.e. 0.33 DM per ton—is, it is true, the lowest of the four items; but this does not, from the cost viewpoint, lessen the importance of the crusher as a cardinal point of reference, inasmuch as it is a long-lasting and virtually unchanging installation as contrasted with the winning, loading and haulage equipment, which is subject to a greater amount of replacement and change. Since there are several units of drilling, loading and haulage equipment to one crusher, such units can successively be replaced without involving interruption of quarry operation.

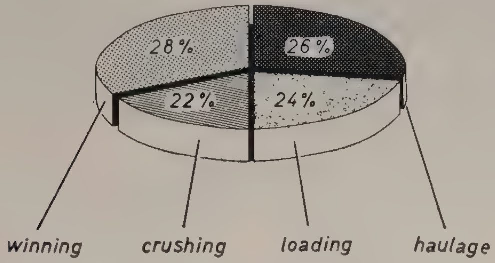


Fig. 3.—Proportions of the cost items in a limestone quarry—winning, loading, haulage and crushing.

In addition to being dependent on the crusher, the loading operations —and therefore also the haulage operations, since the face shovel and the trucks must be interadjusted, i.e. suitably matched in respect of handling capacity—are also dependent on the winning of the material. The height of the quarry face and the drilling and blasting methods significantly affect certain design features of the shovel. The driver and the machine itself must be safeguarded from falling boulders and collapse of the face. With the introduction of blasting with large-diameter bore-holes and the changeover to lower faces (15 to 20 m high), however, control of the working faces is nowadays so effective that it is no longer so important to employ shovels of a certain size. In consequence, winning and loading, as quarrying operations, have become more distinctly separated from each other. Formerly, trimming the face also used to be a task that the shovel had to perform, but nowadays this machine concentrates on loading. Instead of the biting or break-out force of about 60 tons needed for trimming the face, under present conditions a lifting force of, say, about 40 tons applied to the bucket is adequate for a face shovel equipped with a 3·5-m^3 bucket.

Nowadays, the design of the loading equipment is governed solely by the required handling capacity, which, in turn is, limited by the classifying action that the bucket is required to perform. Here again reference can be made to the previous example relating the crusher dimensions and the largest acceptable piece of rock. Blasting with large-diameter holes produces a rock pile of the desired uniform constitution which is immediately suitable for handling by face shovel. Table I shows the grading of a limestone rock pile produced by large-diameter bore-hole blasting.

TABLE I.—*Grading of a limestone rock pile produced by large-diameter bore-hole blasting*

Particle sizes mm	*Proportion % by weight*
0– 200	15
200– 400	20
400– 600	30
600– 800	18
800–1000	10
1000–1400	6
1400–2000	1

In Germany efforts to produce by means of the primary blast the kind of rock pile necessary for uninterrupted loading have, in recent years, been intensified. Even if the rock pile contains only a small number of large boulders, these still necessitate time-consuming clearing away by the shovel and often are of such size that they overload the machine. Their subsequent fragmentation cannot always be appropriately co-ordinated with the loading operations and, if blasting is used for the purpose, this involves costly interruptions of work throughout the quarry. Breaking up boulders by means of the drop-ball method largely obviates this disadvantage and is likely to come into more general use in the future. Boulder-breaking by means of a suitably designed pneumatic hammer attached to the boom of a face shovel is still in the trial stage.

A prerequisite for smooth and uninterrupted loading and for avoiding excessively severe shovel operating conditions is to remove the toe of the face with the primary blast. This obviates the need for secondary blasting and is thus conducive to uninterrupted loading and haulage.

From the foregoing considerations it follows that—and this must be particularly emphasized—loading and haulage are not self-contained operations; rather are they interdependently associated with the whole quarrying process.

HOW LOADING AND HAULAGE AFFECT ONE ANOTHER

Of the external factors so far referred to, those which are bound up with the nature of the quarried deposit cannot be altered. The influencing factors associated with the crushing plant and with the winning operations (producing the rock pile) must also be regarded as predetermined. Within the limits defined by these factors it is, however, necessary to obtain interadjustment of loading and haulage, for only then will economical plant utilization be achieved. The relation between the two spheres of operations can be illustrated by two questions:

(1) How large a capacity should be chosen for the shovel bucket and for the haulage vehicle in order to achieve the predetermined output with the particular loading system concerned, in the following cases:
  (*a*) the trucks back up to the shovel on one side of it, so that there are pauses in loading due to the manoeuvres of the vehicles changing places; and
  (*b*) the trucks drive up, or back up, to the shovel on both sides of it, so that there are no pauses in loading while the vehicles change over?
(2) To attain the predetermined output, how many vehicles having the capacity determined in (1) are needed, having regard to the length of the existing haulage route and to the time taken up by tipping, loading and truck-changing?

The following technical considerations are based on the assumption that the loading and haulage equipment is in constant use. For practical purposes it is necessary to allow for disturbing influences originating in further processing installations and also in the quarry itself. Depending on the operational conditions, such disturbances will be of greater or less significance. For this reason the cost calculation accompanying this paper (Appendix) have been adjusted to practical conditions.

Bucket and truck capacity

If pauses in loading occur, the relationship derived in the Appendix is valid.

$$L_v = \frac{W \times S}{\left(\frac{T \times W}{Q} - t_{Ra}\right) \times F} \qquad (1a)$$

By means of this formula the required shovel bucket capacity L_v (m^3) can be determined from the output Q (tons/h) (which is a predetermined quantity for the particular set of operations considered), the time S (min) needed for the bucket to perform a working cycle, the necessary manoeuvring time for truck-changing t_{Ra} (min), the conversion factors $T = 60$ min/h and $F = 1{\cdot}28$ tons/m^3 (this being based on a bulk density of 1·6 tons/m^3 and an assumed efficiency factor of 80 per cent for filling the bucket, i.e. $0{\cdot}8 \times 1{\cdot}6 = 1{\cdot}28$ tons/m^3), and the truck capacity W (tons) assumed in each particular case. The equation applies only to the loading operation and takes no account of the effect of the length of the haulage route. The underlying assumption is that, apart from the time taken by backing up the trucks to the shovel, there are no pauses in loading. Fig. 4 shows the principle of this situation and indicates the necessary interadjustment of the individual time spans of the haulage cycles.

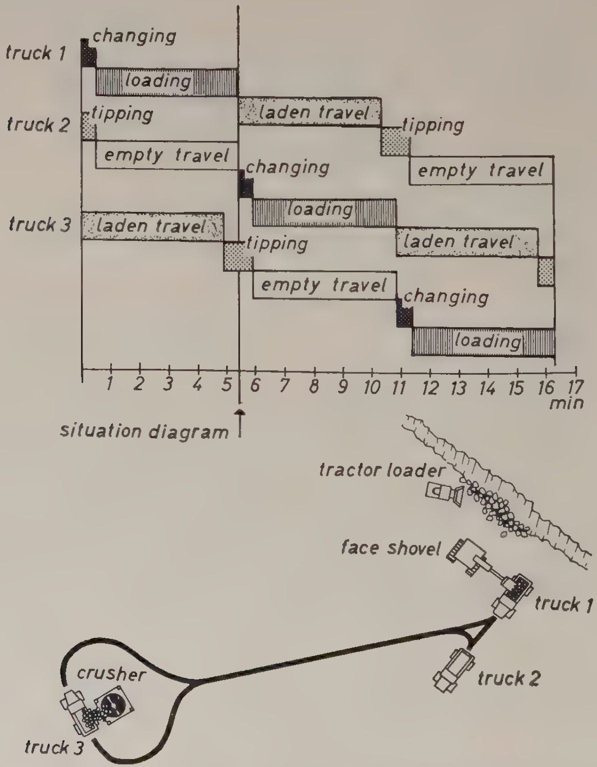


Fig. 4.—Loading of trucks which back up on one side of the face shovel: pauses in loading; arrangement of the haulage cycles.

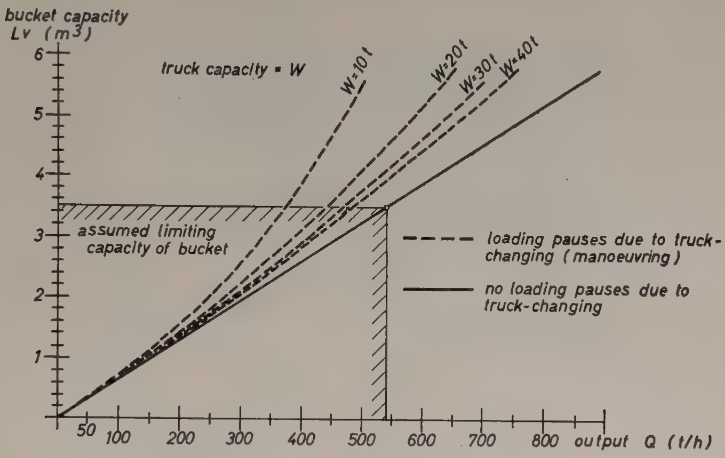


Fig. 5.—Bucket capacity as a function of the chosen truck capacity and the required output.

In conjunction with the output Q to be handled in any particular case, various values can be assumed for the truck capacity W. In addition, for instance, it can be assumed that the bucket working cycle S and the truck-changing time t_{Ra} are each of 0·5 min duration, in accordance with actual practice. With these data the values obtained have been plotted in graph form in Fig. 5. It follows that, for obtaining a certain output, it is possible to choose various sizes of haulage truck, each in combination with a certain size of shovel bucket. The smaller the load-carrying capacity of the truck, the larger must be the bucket capacity in order to offset the increasing total truck-changing time and thus maintain the same output. For progressive decreases in truck capacity the corresponding bucket capacity will have to be varied by appropriate increments. Presupposing constant output, 10-ton trucks may be operated to start with in conjunction with a shovel bucket of appropriate capacity. If a change to 20-ton trucks is made, but the same bucket is still used, the latter will be more disproportionate than if the change were from 20-ton to 30-ton trucks. In the higher output ranges this assumes considerable importance: for example, for an output of 400 tons/h, a change from 10-ton to 20-ton trucks will reduce the requisite bucket capacity by 0·79 m^3, whereas a change from 20-ton to 30-ton trucks will effect a reduction of 0·19 m^3.

It will be seen from equation 1a that the lost time t_{Ra} due to truck-changing is significant. Ideally, however, if there is no lost time the equation assumes the form:

$$L_v = \frac{Q \times S}{T \times F} \tag{2a}$$

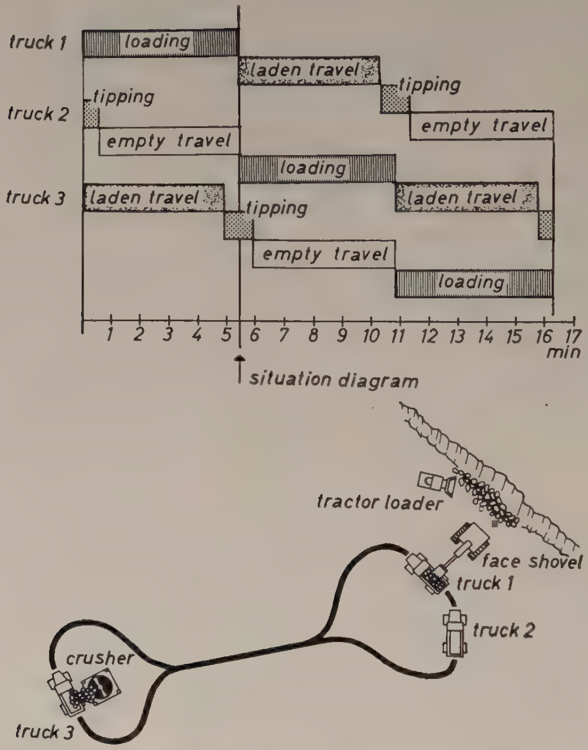


Fig. 6.—Loading of trucks in the case of forward approach to the face shovel: no pauses in loading; arrangement of the haulage cycles.

Figs. 6 and 7 illustrate the principle of the operating procedures corresponding to this. The question as to which is the more favourable method—i.e. forward approach of the trucks as represented in Fig. 6, or backing up on both sides of the face shovel as represented in Fig. 7—will have to be examined according to each particular situation. With the forward approach, the loss of time due to truck-changing is, it is true, eliminated, but account will have to be taken of the longer loading time caused by the shovel having to slew through a larger angle, while the fact that the machine is not stationed perpendicular to the quarry face results in a more unfavourable bucket-filling efficiency factor. On the other hand, backing up to the shovel on both sides results in better shovel utilization. The choice of method must be viewed within the overall framework of loading and haulage operations.

With the elimination of the truck capacity in equation 2a (no pauses in loading) as compared with equation 1a (loading with pauses) it becomes evident that, in the absence of lost time during loading, the

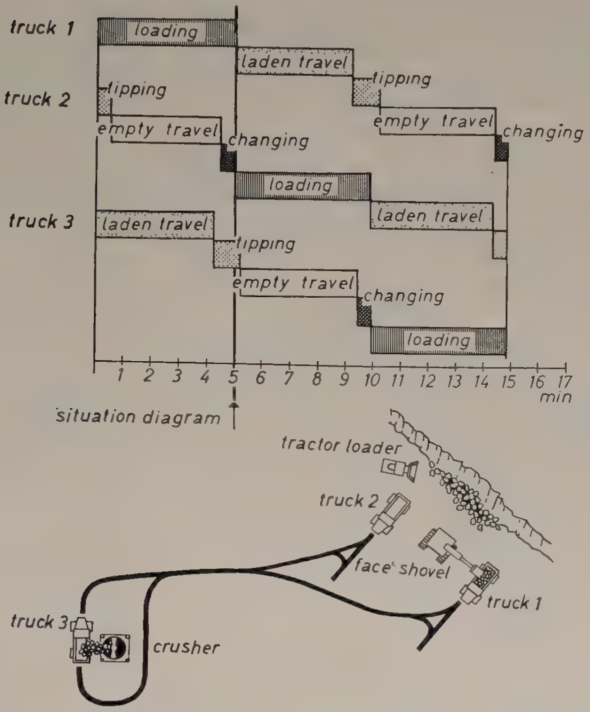


Fig. 7.—Loading of trucks which back up on both sides of the face shovel: no pauses in loading; arrangement of the haulage cycles.

truck capacity has no effect on the attainment of a certain output. In that case only the capacity of the bucket determines the magnitude of the output Q. Assuming a bucket working cycle $S = 0{\cdot}5$ min, as before, certain bucket capacity values are obtained which have also been represented graphically in Fig. 5. As equation 2a already indicates, there is a simple linear relation; this can be derived from the curves corresponding to equation 1a (which are associated with pauses in the loading operation) as an extension of their range. For if, in that formula, the truck capacity increases and the sum of the loading pauses consequently decreases, the curves will more and more closely resemble straight lines until, in the ideal case, they attain linearity.

The limit applicable to the family of curves is significant. It is governed by the installed crusher capacity, inasmuch as the latter determines the upper limiting value for the capacity of the face-shovel bucket and thereby determines the output that is still just attainable with a particular size of haulage truck. Thus an assumed limiting bucket capacity of, for example, 3·5 m^3 will, if no loss of time is incurred in truck-changing manoeuvres, in this case enable an output of up to 540 tons/h to be attained.

Number of trucks

A certain number of haulage vehicles must be available in order to avoid pauses due to lack of empty trucks in readiness for loading during the periods of travelling, tipping and truck-changing. The haulage facilities must cope with the same output Q as that handled by the shovel in loading the trucks.

On this assumption, the number of trucks needed is:

$$N = \frac{Q\left(\frac{2a}{v} + \frac{W \times S}{L_f} + t_{Ra} + t_{Br}\right)}{W \times T} \tag{3}$$

This equation, like equation 1a, takes account of the fact that pauses in loading occur as a result of backing up the trucks. Its derivation is given in the Appendix. Additional symbols introduced in this equation are: the haulage distance a (km), the speed v (km/h), the bucket load (bucket contents) L_f, and the time t_{Br} (min) required for tipping the contents of the vehicle at the crusher. The bucket load L_f is given by the following formula:

$$L_f = \frac{W \times S}{\frac{T \times W}{Q} - t_{Ra}} \tag{1b}$$

(see Appendix: L_f represents the quantity of material, in tons, scooped up from the rock pile; the bucket capacity L_v is expressed in m³).

On the basis of empirical values obtained in actual practice, an average speed $v = 18$ km/h is assumed for the evaluation of equation 3. The truck-changing time t_{Ra} and the bucket working cycle S will each again be taken as 0·5 min, and the time spent at the crusher is estimated at 1 min.

For the adopted initial values, the relationships between output, haulage distance, truck capacity, number of trucks, and bucket load, are represented graphically in Fig. 8. Each of the curves plotted in the diagram relates to a certain output Q and to a certain haulage distance a, and for this set of conditions the requisite number of trucks—depending on the truck capacity adopted, for which a choice between 10, 20, 30 and 40 tons has been assumed—can be read from that curve. The general trend of the family of curves is evident: for constant haulage distance and increasing output, for constant output and increasing haulage distance, and also, of course, if both these quantities increase, the requisite number of trucks will become larger (assuming the capacity per truck to remain the same).

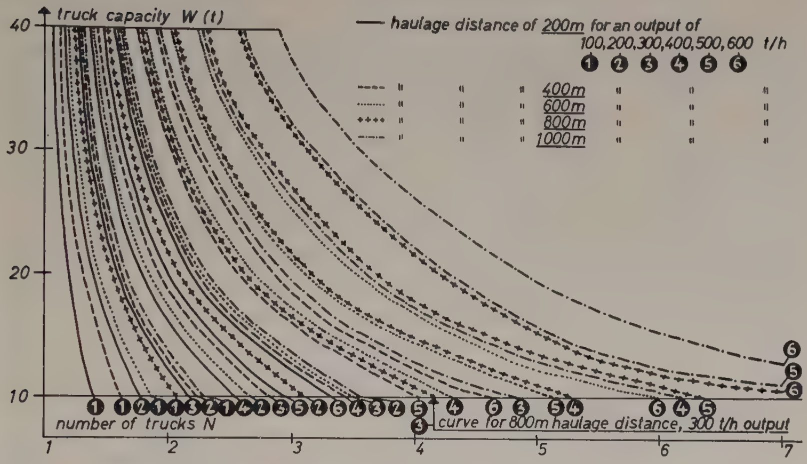


Fig. 8.—Requisite number of trucks as a function of output, truck capacity and haulage distance.

From Fig. 8 it is also apparent that, for equal haulage distance and output, different overall truck capacities (total tonnage load-carrying capacity) are necessary, depending on the capacity of the type of truck employed. For example, if the haulage distance is 800 m and the output is 300 tons/h, the total tonnage capacity required is either 41·7 tons with 10-ton trucks (4·17 × 10 tons), or 51·6 tons with 20-ton trucks (2·58 × 20 tons), or 61·5 tons with 30-ton trucks (2·05 × 30 tons), or 71·6 tons with 40-ton trucks (1·79 × 40 tons). The differences are easily explained: comparing, say, a 20-ton with a 30-ton truck, the former can start its haulage trip as soon as it has been loaded with 20 tons of rock, whereas the latter vehicle must spend additional time at the shovel for taking in a further 10 tons, so that here a certain amount of haulage capacity is still lying idle which, in the other case, is already performing the transport function. The following general inference can therefore be drawn: continuous haulage with a smaller size of truck enables the haulage operations to be maintained with a smaller total vehicle tonnage capacity.

Fig. 8 is a set of curves representing a number of output/haulage combinations. If the calculations are suitably extended, curves for intermediate and adjacent regions can readily be obtained, so that data applicable to any loading–haulage situation can be determined. Of course, the initial values assumed here can be appropriately varied in any particular case, but the basic results remain valid.

Presupposing the operating methods which involve no loading pauses (Figs. 6 and 7), equation 3 also holds good for the procedure in which the trucks alternately back up on each side of the face shovel, but the bucket load L_f corresponding to equation 2a (no pauses) should be substituted. With forward approach of the trucks to the shovel, however, equation 3 is valid only without the term t_{Ra}, because now there are no losses of time due to manoeuvring in the individual truck-working cycle. In this case, too, the bucket load corresponding to equation 2a should be taken into account.

A numerical evaluation of these operating methods is not presented here. The above discussion provides an example of a fundamental consideration of the problem.

The results derived from the graphs must be modified to suit practical conditions, as the haulage units, which have been numerically determined to within hundredths, have to be rounded off to a greater or lesser degree, and this in turn will justify the choice of one or other type of vehicle. For example, a truck which is only one-tenth needed for performing the haulage function, represents dead capital insofar as the other nine-tenths is concerned.

Loading and haulage costs

Cost is ultimately the decisive issue in all technical considerations. In the following a comparative cost calculation is given with reference

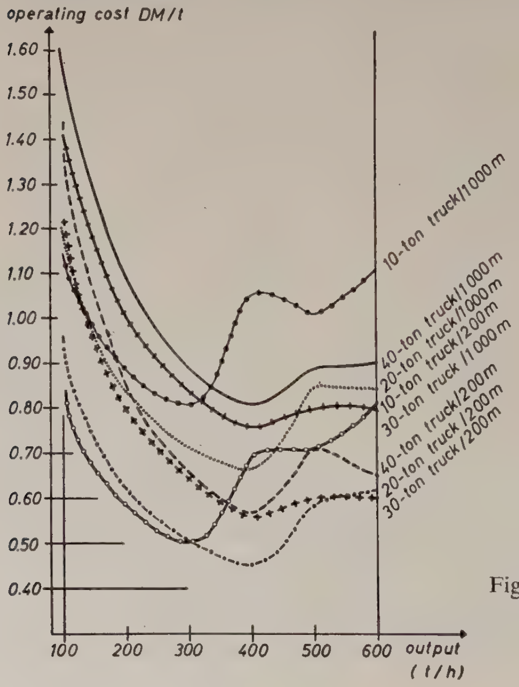


Fig. 9.—Loading and haulage costs as a function of output, truck capacity and haulage distance.

to the actual basic costs associated with the use of the appropriate truck and shovel combinations that have been theoretically determined in the examination of the problem already made.

Fig. 9 gives the result of the comparative calculation for a haulage distance of 200 m and 1000 m respectively. Although to go into the details of the calculation are outside the scope of this paper, the essential basic assumptions are: an output of 10 000 tons/day, for a 16-hour working day with an effective working time of 12 hours (breakdowns, blasting, change of shifts), with reserve capacity of 20 per cent for face shovels and of 25 per cent for haulage vehicles. The cost calculation is subdivided to consider various cases where the 10 000 tons/day output is yielded by loading points at which the material is loaded at rates of 100, 200, 300, 400, 500 and 600 tons/h respectively.

There is no reason for seeking any regular pattern of curves in the cost diagram, because the necessary rounding off of the theoretically determined numbers of trucks and face shovels results in more or less excessive reserve capacities in individual cases. Also, with increasing size of plant, the initial cost is not quite proportional to the size of the units, and there is unevenness.

Nevertheless, the cost diagram does exhibit a general trend: with increasing distance the vehicle with the larger load-carrying capacity is already more economical at lower output. For example, for a haulage distance of 1000 m and a haulage rate of 200 tons/h the 20-ton truck, with the appropriate shovel, operates more economically (0.84 DM/ton) than the 10-ton truck (0.88 DM/ton). On the other hand, for a haulage distance of 200 m the 20-ton truck, with the appropriate shovel, becomes more advantageous only from a haulage rate of 300 tons/h upwards.

The breakdown of the operating costs, as determined here, into different cost categories is of particular interest. With regard to the loading costs (Fig. 10), in investigations carried out for a limestone quarry the percentage cost breakdown was as follows: power and

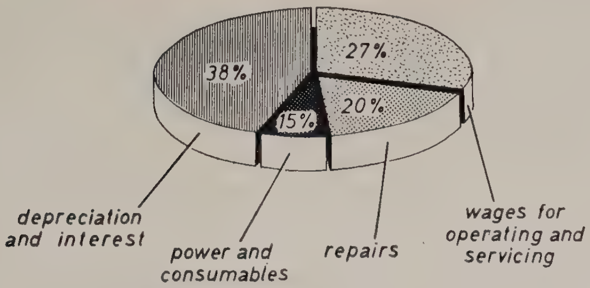


Fig. 10.—Proportions of the cost items in loading with face shovels of 3·5 m^3 capacity.

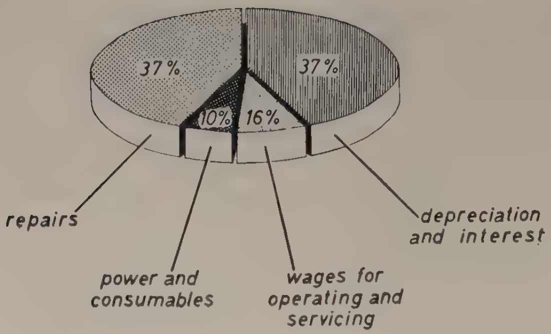


Fig. 11.—Proportions of the cost items in haulage with 40-ton semi-trailer vehicles.

consumables 15 per cent; repairs 20 per cent; wages for operating and servicing 27 per cent; depreciation (15 years) and payment of interest (5 per cent of half the initial outlay) 38 per cent. The cost breakdown for haulage (Fig. 11) shows a similar pattern of increasing proportions: fuel and consumables 10 per cent; wages for operating and servicing 16; depreciation (5 years) and payment of interest (5 per cent of half the initial outlay) 37; repairs, including tyres, 37.

LOADING AND HAULAGE EQUIPMENT

The present considerations have, so far, been more in the nature of a statement of general principles. In the following sections, however, the loading and haulage equipment will be more particularly considered, with examples where possible.

Loading Equipment

The face shovel

The face shovel is the principal loading machine for quarry use. While being aware of the many problems of design, only those components which play a particularly significant part in loading operations will be discussed.

The design of the bucket, or dipper, is of major importance in its contribution to the effective work performed. Fig. 2 shows the form of bucket that has proved most efficient from this point of view. The curved projecting fore part of the bucket, and the shape of the teeth, help the bucket to penetrate into the rock pile. Whereas buckets of earlier design only attained filling efficiency factors of 60–70 per cent (referred to the level bucketful), this bucket can reach 90 per cent when operating in a good rock pile.

The reason why the width of the bucket (1700 mm) is greater than the depth (1440 mm) is that the bucket width determines the width of 'cut' and thus the quantity of material that is scooped up in the course of the available length of travel through the rock pile. The depth of the bucket, however, does not have the same influence on the amount of material scooped. A further feature is the gradual progressive increase in depth from the intake to the discharge section of the bucket, designed to prevent material being jammed when the bucket is emptied.

The interstices between the teeth are now preferably made 250 mm wide, which is substantially less than the former width of about 400 mm. This development should be viewed in relation to the now predominant use of large-diameter bore-holes for blasting, which produce better fragmentation and smaller pieces than tunnel blasting. The rock pile produced is more uniform in composition and is not so tightly packed, so that the bucket can operate more efficiently with closer spacing of the teeth.

The teeth are more severely affected by abrasive wear than are the other parts of the bucket. Operating data for limestone show that up to about 300 000 tons of rock can be loaded before the teeth are worn out. This high loading performance is, however, associated with a change in the internal structure of the metal, making subsequent build-up welding or the repeated application of such welding not very effective. It is rather different with types of rock that cause the teeth to wear more rapidly, as in this case the structure of the metal will not, by that time, have become impaired, so that build-up welding is entirely to be recommended. For instance in dolomite rock, bucket teeth with build-up welding can be expected to have a service life corresponding to about 80 000 tons of rock loaded. Under the severe conditions of a quartzite quarry there is a corresponding life of only 24 000. Efforts are, however, being made to counteract tooth wear by means of fit-on pieces, and thus avoid expensive welding.

The position of the pivot of the dipper stick, or handle, is important, being determined on the one hand by the fact that the working radius is larger in proportion the higher up the pivot is placed, but on the other by the dumping height of the bucket. This height should be such that when the discharge section of the bucket is horizontal the bucket contents do not fall from so great a height as to impose excessively severe conditions of service on the haulage vehicles.

The high stresses to which many components are subjected make it necessary to use suitable materials. For example, the bucket teeth and front wall of the bucket are usually made of austenitic manganese cast steel; a material often used for crawler track links is chromium alloy cast steel. In addition to the choice of material, the heat treatment applied gives higher strengths to entire components or special resistance to abrasion.

Improved components have been developed recently for performing the many movements of the various parts of a face shovel, often involving considerable accelerations and decelerations. The adoption of separate drives for the main motions—hoisting, crowding, slewing and travelling—has resulted in the elimination of elaborate transmission systems. The electric drives are generally installed, as far as possible, at the actual point where the power has to be applied. Also, the initiation of the motions by means of electric controls has facilitated machine operation and has thus substantially increased shovel efficiency.

As regards suitable types of drive a distinction must be made between the asynchronous motors fed with three-phase alternating current and the various modifications of the Ward-Leonard system. The use of inductive or electronic control in connection with the latter has proved effective. These methods provide better adaptability, resulting in increased output and flexible operation, and thus reduce wear and tear of the machine as a whole.

Present-day face-shovel motors are, of course, designed for continuous operation. With rail-mounted haulage it is still possible to equip the shovels with motors rated for 40–50 per cent intermittent duty; with rubber-tyred haulage, however, shovel motors of that kind are not suitable because of the more rapid and uninterrupted removal of the loaded material.

In order to utilize the advantages of high-voltage electric power, there has, in part, been a changeover from the usual low-voltage supply (380 V or 500 V) to 4000 V. The transformer station supplying the power is located at a distance of 150–200 m from the face shovel or may, indeed, be mounted on the shovel itself so as to minimize voltage drops.

The use of electromagnetic clutches calls for special mention. The principle of this form of transmission—known as the 'magnetorque'—is that a magnetic field of force is produced whose strength is varied so as to transmit a torque of corresponding magnitude. With a system of this kind, control of any particular operation is effected through the agency of the magnetorque clutch, while the motor runs continuously at constant speed.

The application of electric power has also been extended to braking. As a general principle, in modern face shovels mechanical braking is used only for securing the travelling gear of the machine when it is stationary, whereas the various motions performed by the machine in operation are electrically braked by means of the control equipment. As a result, the mechanical construction of the shovel is simplified and greater operational reliability is ensured.

This outline of the present position of face-shovel engineering shows that the mechanical principles of the operating motions of the machine have not undergone any change in recent years. Well-tried fundamental features have been improved in detail. Development of a more intensive kind is presented by the drive systems in the modern shovel, however, in that these are more specifically assigned their particular tasks, each motion being powered by its own individual set of drive equipment. Improved control methods and, as a result, smooth and uninterrupted operation, have lessened the drawback of interrupted scooping up of material from the rock pile. Efforts should still be made to apply to quarrying the principle of the continuous loader, e.g. that of the Joy loader or the rotary-bucket excavator. The moving of heavy dead loads, which is a disadvantage of the type of shovel described here, would then be largely eliminated.

Tractor loaders

Ancillary equipment which supplements the loading work performed by the face shovel is indispensable in present-day quarrying. These machines have the task of collecting scattered material and of building up the toe of the rock pile—both tasks which used to be performed by the face shovel, but which were liable to cause breakdowns and necessitate repairs. In many cases it is best to choose a machine that is suitable for both grading (levelling) and loading; it is then possible to use the machine to a limited extent for loading operations. There are no indications, however, that the face shovel is likely to be superseded by a machine of this kind.

Whereas this ancillary equipment has hitherto almost exclusively been of the crawler-mounted variety, the advantages of rubber-tyred machines fitted with non-skid chains have more recently been turned increasingly to good account. Manoeuvrability and speed are characteristic features of these vehicles, speed being especially important in their availability for a wide range of tasks. They are also advantageous from the viewpoint of cost, a comparison based on the use of both types of machine in a storage yard for rock showing the running cost of the 150-hp crawler vehicle as 36.69 DM per hour, but 30.50 DM per hour for the wheel tractor loader. The main reason for this significant difference is that repairs are simpler to carry out with the latter machine. Whereas the travelling gear of a crawler-mounted machine takes about 300 man-hours to renew, the tyres of a wheeled machine can be changed in only about 8 hours. A similar difference in costs would presumably also be found under quarry operating conditions.

Haulage Equipment

Haulage by rubber-tyred vehicles has now become standard practice in quarries because it is more economical. For instance in the case of a section of quarry with rail-mounted haulage over a distance of about 700 m the cost of loading and haulage comes to 0.67 + 0.59 = 1.26 DM per ton, whereas a change to haulage with rubber-tyred 35-ton vehicles will, it is anticipated, cost, under prevailing conditions, 0.34

+ 0.40 = 0.74 DM/ton, i.e. a reduction of 41 per cent brought about by the better utilization of the loading equipment.

Only where large deposits of considerable extent and very large quantities of material have to be conveyed could haulage by means of rail-mounted vehicles be expected to be economically advantageous.

It will usually be possible, in the quarry, to avoid incurring any special expenditure on road construction, except perhaps on the loop road around the crusher, or other road sections similarly subjected to severe loading, where a hard surfacing would be advantageous. It is, however, essential to keep the whole of the haulage route in good repair, and the motor grader has proved very efficient for this purpose, pushing fallen lumps of rock off the road and levelling the surface when it has been made uneven by the passage of vehicles.

As regards gradients of haulage roads, experience has shown that both for uphill and downhill routes a gradient of 1 in 10 should not be exceeded. This is determined by considerations of safety with regard to vehicle control and the danger of lumps of rock falling off the vehicle. On uphill gradients there is the additional requirement that the vehicles must be able to restart and get moving from the stationary position. In this connection, difficulties do not arise from any inadequacy of engine power or lack of grip on the ground, but from the nature of the power transmission to the driving axle. Smooth starting, as provided by torque converters, is not possible with the aid of simple friction clutches.

There are indications that it is an advantage, as a general rule, to provide vehicles of more than 30-ton payload with a torque converter. While this, and the more powerful engines necessary, do substantially increase the capital expenditure, the additional initial outlay is offset by higher output due to higher speed of travel, especially on uphill and downhill runs.

The engine power must be suited to the road conditions. Power requirements for various types of 40-ton vehicles are indicated in Table II.

TABLE II.—*Engine power requirements for 40-ton vehicles operating on horizontal and inclined haulage roads*

Haulage on horizontal stretches or on uphill or downhill gradients up to 3 in 100	*Uphill, not exceeding 1 in 10*	*Downhill, not exceeding 1 in 10*
(1) Rear-dump or side-dump truck with trailer (300 hp)	(1) Rear-dump or side-dump truck (over 350 hp)	(1) Rear-dump or side-dump truck (over 350 hp)
(2) Semi-trailer vehicle (300 hp)	(2) Semi-trailer vehicle (over 350 hp)	

Owing to lack of adequate experience, it is not yet possible to make any definite statements on engines equipped with exhaust-gas turbo-supercharging. It seems likely, however, that—like every technical over-development—it will result in diminished service life of the engine.

Axles fitted with planetary reduction gearing and the advantages associated with this—namely, that the other transmission elements that precede it can be of simpler design and that the requisite final torque is developed only at the point where it actually has to be applied—are now standard features.

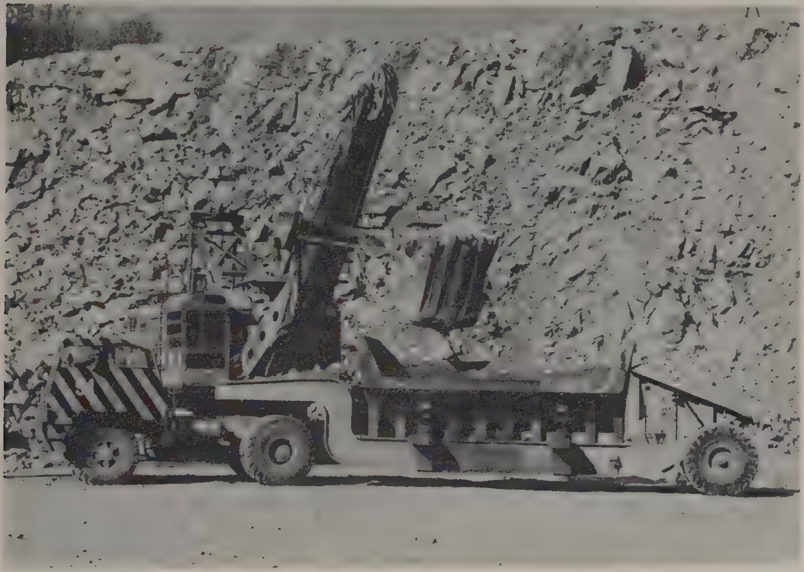


Fig. 12.—40-ton semi-trailer haulage vehicle.

Of major importance are problems relating to the design and construction of the dump body or hopper of the vehicle. The nature of the tipping gear is important. This may be either mounted on the vehicle itself and hydraulically operated or stationary at a fixed discharging point and mechanically operated. The latter type of installation may yield significant advantages in terms of cost. An annual operating cost of 11 500 DM was recorded for five 40-ton semi-trailer vehicles (Fig. 12) served by one stationary tipping winch installation (initial cost 60 000 DM, not including the supporting gantry frame, which had already been needed for erecting the crusher). If the vehicles had been equipped with individual hydraulic tipping gear (each set costing 16 490 DM), the anticipated total annual cost would have been around 47 000 DM. The saving effected by the solution chosen in this

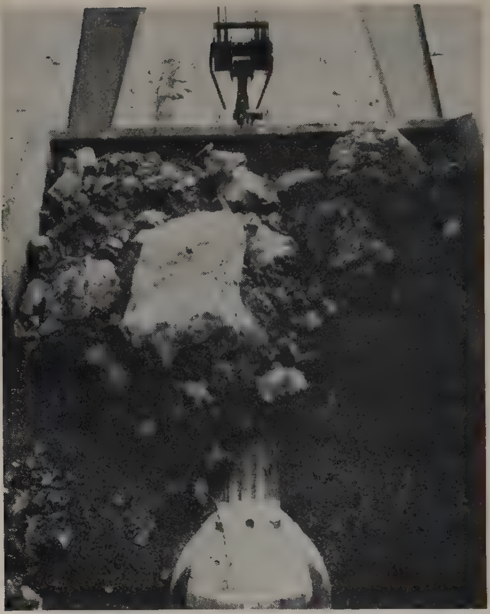


Fig. 13.—40-ton dump body being emptied with the aid of a stationary tipping winch installation.

case is therefore very striking. Fig. 13 shows a dump body of 40-ton capacity being emptied by a stationary tipping winch installation.

So far, no particular typical shape has yet been evolved for the dump body. The varying conditions of service imposed by loading, travel, tipping and general rough treatment often make one form of construction or another appear the best, but the determining factor with regard to body shape is that it must provide a sufficiently large opening for expeditious loading and that, on tipping the contents, it can be quickly and completely emptied. If this technical requirement is fulfilled, it must also be ensured that it really is utilized. Experience has shown, in fact, that the men operating the crusher—e.g. a gyratory crusher which is not equipped with a feeder—often have a preference for intermittent tipping so as to minimize the amount of manual labour involved in clearing the crusher shaft before restarting the machine in the event of unforeseen failure of the drive.

Disorders of this kind have led to the adoption of a second drive which, through suitable gearing, develops a high torque at very low speed of rotation of the drive shaft and thus enables the crusher to clear and release itself. This will, it is anticipated, overcome the difficulties.

High-pressure tyres (4·5–6·0 atm gauge pressure), medium-pressure tyres (3·0–4·5 atm gauge pressure) and low-pressure tyres have hitherto all been used on haulage vehicles. Experience in a limestone quarry has led to a preference for medium-pressure tyres, since with such tyres any excessive 'churning' is avoided while they nevertheless have some degree of flexibility for coping with irregularities of the road surface. Special features, as compared with ordinary tyres, are a thickening of the side shoulders as a protection against being ripped open by sharp-edged stones. To prevent damage to the inner fabric by stones that have penetrated, a recent development has been the insertion of a layer of steel chips between the fabric and the breaker overlying it. In the limestone quarry referred to, service lives of up to 5500 working hours were obtained with medium-pressure tyres of conventional construction.

The need for keeping a check on performance and costs in haulage means that the quantities handled and conveyed have to be determined as a normal regular feature. Automatic weighing installations have been found very suitable for the purpose as they provide a clear survey at all times and enable the situation to be assessed at short notice.

Concerning the haulage of non-utilizable materials, it is recommended that, generally speaking, smaller vehicles should be used for the selective removal of impurities. In choosing these vehicles it should be remembered that it is often necessary to climb gradients steeper than 1 in 10 and that difficult riding conditions are liable to be encountered on the journey to the tipping area. For these reasons vehicles of the all-wheel-drive type are suitable, equipped with their own hydraulic tipping gear as a matter of course, as the tipping points shift about. It is preferable to use end-tipping (rear-dump) vehicles. Vehicles of 10- or 20-ton payload capacity are used under the conditions envisaged, and for the gradients of up to 16 in 100 to be negotiated, the engine power required is 120–180 hp for the 10-ton and 180–250 hp for the 20-ton vehicles.

PROTECTION OF PERSONNEL

In quarrying there are many potential dangers to personnel from the natural rock formation, from the working method applied and from the machines used. To recognize and counteract these dangers is essential at every stage of quarry operation. Successful working is possible only if human safety takes precedence over material considerations. This is reflected in the numerous safety precautions applied in the conduct of the quarrying operations and in the design and handling of the mechanical plant. In addition, there are danger warnings in the form of safety signs and colours. Good organization in the co-ordination of the many different activities is also conducive to safety. In the last resort, however, no safeguards can ever be completely effective without personal caution and prudence. Real success can be achieved only by obeying the rule 'Safety First'.

Appendix

(A) Bucket and truck capacity for predetermined output

The hourly output Q at a loading point is

$$Q = W \times N \times \frac{T}{tn}$$

Furthermore

$$N = \frac{tn}{t_{La} + T_{Ra}} \text{ and } t_{La} = \frac{W}{L_f} \times S$$

where W = truck capacity (tons)
N = number of trucks serving the loading point, assuming continuous removal of material by haulage
T = conversion factor: 60 (min/h)
tn = time required for performing a haulage cycle (min)
T/tn = number of haulage cycles of a truck per hour
t_{La} = time for loading a truck (min)
t_{Ra} = time for backing up a truck to the shovel (min)
L_f = bucket load (contents) (tons)
S = time required by bucket for performing a working cycle.

By combining the above, the following are obtained:

$$Q = \frac{W \times T}{\dfrac{W \times S}{L_f} + t_{Ra}}$$

$$L_f = \frac{W \times S}{\dfrac{T \times W}{Q} - t_{Ra}} \tag{1b}$$

To calculate the bucket capacity L_v from the bucket load L_f this must be divided by a factor $F = 0{\cdot}8 \times 1{\cdot}6$ tons/m^3 $= 1{\cdot}28$ tons/m^3 (efficiency factor for bucket filling = 80 per cent; 1 m^3 of rock pile = 1·6 tons).

$$L_v = \frac{W \times S}{\left(\dfrac{T \times W}{Q} - t_{Ra}\right) \times F} \tag{1a}$$

If the truck-changing time t_{Ra} is dropped, then

$$L_v = \frac{W \times Q}{T \times F} \tag{2a}$$

(B) Requisite number of trucks for predetermined output

The haulage system must cope with the hourly output Q:

$$Q = W \times N \times \frac{T}{tn}$$

In addition

$$tn = t_w + t_{La} + t_{Ra} + t_{Br}$$

$$t_w = \frac{2a}{v}$$

$$t_{La} = \frac{W}{L_f} \times S$$

where the additionally introduced symbols have the following meaning:

t_w = time for travelling to and returning from the crusher (min)
t_{Br} = time spent at crusher (min)
a = distance from loading point to crusher (km)
v = average speed of vehicle travelling laden and empty (km/h).

Combining the above expressions gives

$$Q = \frac{W \times T \times N}{\frac{2a}{v} + \frac{W \times S}{L_f} + t_{Ra} + t_{Br}}$$

or

$$N = \frac{Q\left(\frac{2a}{v} + \frac{W \times S}{L_f} + t_{Ra} + t_{Br}\right)}{W \times T} \tag{3}$$

REFERENCES

1. STUMPF, K. Abstimmung zwischen Sprengmethode, Baggerabmessungen, Fördermittel und Brechergrössen im Steinbruch. *Zement-Kalk-Gips*, **7**, no. 4 1954, 124–30.

2. STUMPF, K. Baggergrössen, Leistungen und Kosten. *Zement-Kalk-Gips*, **10**, no. 12 1957, 498–500.

622.619.4:622.341.1(943)
338.58:622.6:622.271.3

Loading and Transport: Experience at Mount Morgan, Queensland

GLENISTER SHEIL, B.Sc., B.C.E., B.M.E.(*Melbourne*), M.I.M.M., M.Aus.I.M.M.
Technical Director, Mount Morgan Limited

SYNOPSIS

The paper describes the loading and transport of 90 000 tons/week of ore and waste, of widely varying physical characteristics, over distances up to 1½ miles and on grades up to 1 in 7½, from the confined opencut of Mount Morgan Ltd., Central Queensland, Australia.

The Company uses 14 shovels with bucket capacities from 1½ to 5 yd^3, and 2 front-end loaders with bucket capacities of 1½ yd^3.

The operation and travelling of these shovels on narrow benches and inclines as steep as 1 in 7½ are described, including lowering them 100 ft vertically down a face sloping 1 in 1. Unit costs of loading for the various machines are given.

Transport is carried out by a fleet of 15-ton payload diesel trucks, of which half are three-axle type (like the Diamond T truck) and the other half two-axle type (like the Euclid). The comparative performance of these types of trucks is discussed, as is the comparative performance of United States, Australian, German, Belgian, French and Japanese tyres. Unit costs of transport are quoted.

Engineering maintenance and rebuilding of opencut plant for loading, bulldozing and transport are featured, together with relevant costs.

In 1882 the first ore from the capping of the Mount Morgan orebody was treated. In 1886 the Mount Morgan Gold Mining Co., Ltd., was registered in Rockhampton with a capital of £1 000 000. Until 1891 exploratory driving only was carried out, but in that year more vigorous exploratory work resulted in the Crown and the Sugarloaf shaft being sunk and the Linda Gully tunnel being driven into the mountain side at a point 460 ft below the original summit. Undergound stoping was begun in the upper levels of the mine. Then, because of the discovery in 1904 of the continuation of ore in depth, the Main haulage and Linda haulage were put in (Fig. 1).

During the early 1920s underground mining gradually became unprofitable owing to rising costs and the falling grade of ore. A review of the operation in 1925 brought forward the suggestion that large-scale opencutting, developed from and using existing mine levels and hoisting facilities, be adopted. The schemes proposed would make up to 7 000 000 tons of ore available. An American engineer considered an

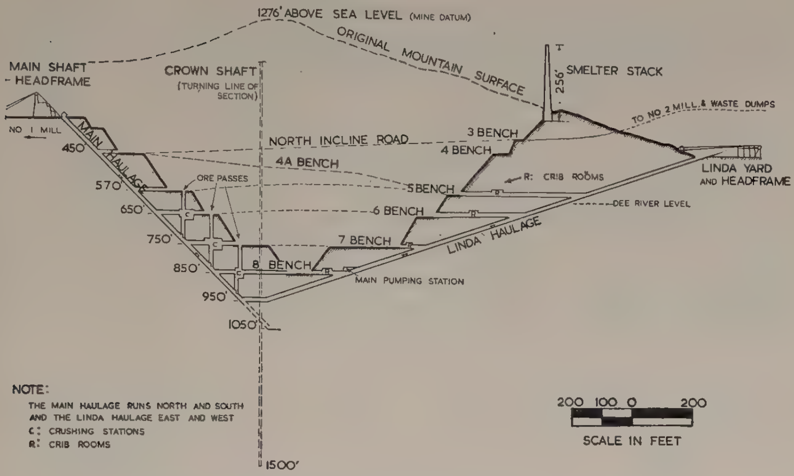


Fig. 1.—Diagrammatic cross-section of the opencut.

opencut feasible, but the ore available would not warrant the expenditure required to start electric shovel operations.

Two schemes were proposed. (1) The engineer envisaged short bench intervals and spiral rail haulage to the surface from the 750-ft level, with a final overall batter of less than 45°. This scheme involved the immediate removal of overburden for development by spiral railway, preceding mining of ore. (2) An alternative method was to reach the 850-ft level with a 52° overall batter. Material broken on each level below the 450-ft was to be transported to crushing stations on these levels, to be crushed before being hoisted to the surface in either the Main or the Linda shaft. This scheme involved the immediate production of ore concurrently with the removal of overburden. This scheme was not accepted by the Company at that time, possibly because of the engineer's report, but it came into use in 1938.

The first scheme involved the removal of 34 400 000 tons of overburden to recover 6 000 000 tons of ore averaging 1·25 per cent copper with 3·89 dwt of gold per ton; the second meant moving 21 000 000 tons of overburden and opening up 7 000 000 tons of ore, giving 1·72 per cent copper 4·37 dwt of gold per ton. This scheme is called the '7 000 000 tons Scheme'.

Opencut operations were begun in 1932, but the rate of development was slow until after the Second World War. To the end of the year 1961–62 opencut mining has produced 1 607 972 oz gold and 136 443 tons copper from 21 989 600 tons of ore, which were mined with the removal of 45 820 400 tons of overburden. Figs. 1 and 2 depict the size of the opencut today, which covers an area of 74 acres, the bottom level being 850 ft below the original mountain top.

GEOLOGY

The orebody occurs within a roof pendant of metamorphosed Devonian and Carboniferous grits, tuffs, lavas, cherts and limestones, the wide pendant being overlain unconformably by flat-lying Mesozoic sandstones. Rock deformation has resulted in the development of intense jointing and shearing of the mine rocks and the obliteration of much of the original bedding and flow structures.

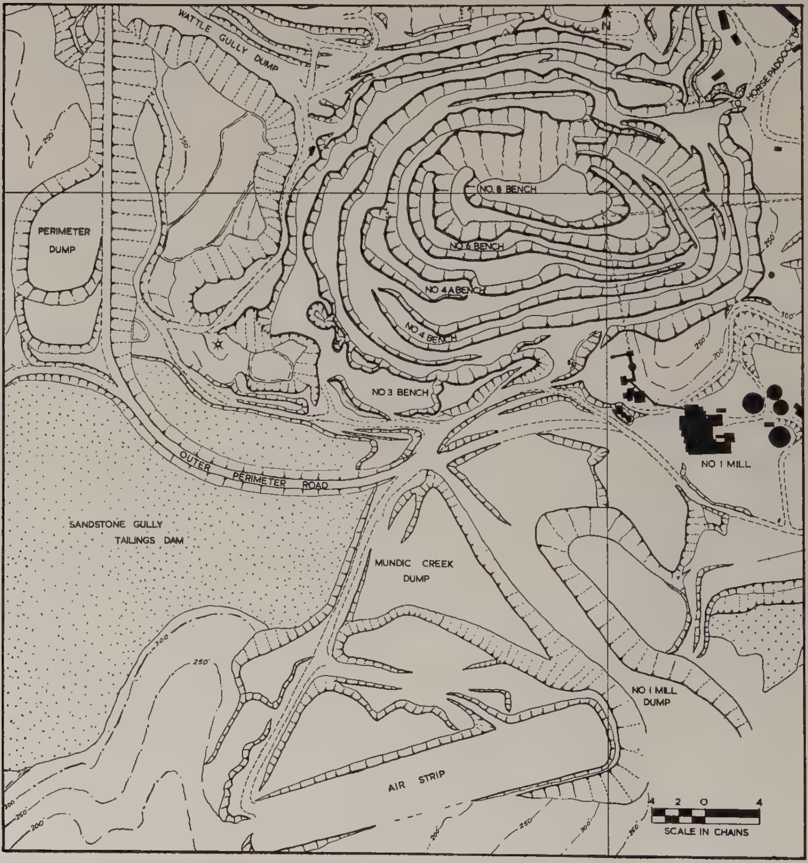


Fig. 2.—The opencut plan.

The deposit consists of the Mount Morgan orebody, the main outcropping orebody, and the contiguous Sugarloaf orebody delineated in 1951. Ore material is a hard quartz–pyrite rock containing chalcopyrite as the principal copper ore mineral.

There are three fault systems, two of which—the Slide and the Linda Fault—have an important bearing on the disposition of the orebody. The Slide, striking NE–SW and dipping at 65° to the SE, causes some difficulty in mining. The Linda Fault strikes NW–SE and dips at 25° to the SW underlying the orebody. The third fault, the Footwall, forms the eastern margin of the orebody.

Intruding the sediments, the orebody and the granitic rocks is a complex system of pre- and post-ore dykes of intermediate and basic composition ranging from a few inches to 60 ft in thickness.

Characteristics of mine rocks

(1) Gold–copper ore—gold and chalcopyrite in a pyrite–quartz gangue; heavy, blocky and abrasive, concentrated at No. 1 mill;

(2) Low-grade ore—(*a*) containing pyrrhotite; heavy, blocky and abrasive; (*b*) sulphides; heavy and abrasive, concentrated at No. 2 mill;

(3) Feldspar porphyrite dyke—(up to 60 ft in width); heavy and blocky;

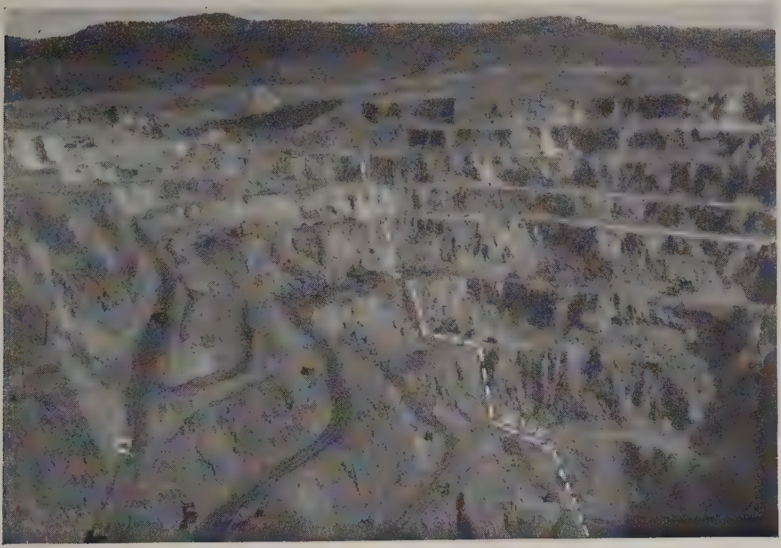


Fig. 3.—The opencut looking west showing the Slide and the variation of material within the mine.

(4) Felsite—the principal host rock in the mine; easy to handle;

(5) Oxidized siliceous outcrop—easy to handle;

(6) Stope fill—(*a*) waste material from the original top of the mountain; (*b*) mine tailings; this material would be easy to handle if a large quantity of mine timber did not have to be removed first.

Fig. 3 shows the Slide and the variation of material within the mine.

DRILLING AND BLASTING

For efficient loading and transportation the material must be suitably fragmented at the lowest possible cost, that is with the largest possible quantum of primary blasting and as little secondary reduction as possible. Conditions for drilling and blasting in both ore and overburden are difficult, however, owing to the jointed and faulted nature of the rocks, the presence of influent seepage water in the western side of the opencut and the effect of fire and subsidence, together with lack of homogeneity of the stoped areas.

These complex conditions and the need to mine selectively overburden, stope filling, No. 1 mill ore, No. 2 mill ore and dyke material within the orebody, make necessary a variety of drilling and blasting techniques.

Primary drilling is usually confined to down holes from 30 to 120 ft in depth, although side holes, breast holes and toe holes up to 18 ft in length are also drilled. Drill-hole sizes are 2 in. in diameter for horizontal holes, 3½ in. for down holes in ore and hard overburden, and 6½ in. or 7 in. for down holes in softer overburden. From trials of numerous firing patterns, maximum and minimum burdens and spacings for down holes used throughout the mine have been derived. Maximum and minimum burdens and spacings for hard overburden and quartz–pyrite ore are shown in Table I.

TABLE I

Drill	*Bit dia.*	*Maximum burden × spacing*	*Minimum burden × spacing*
	in.	ft	ft
Heavy track	6½	20 × 24	17 × 17
Crawl-IR	3½	12 × 12	9 × 10
Diamond drill	Ex	7 × 8	6 × 6

For all holes above 3 in. in diameter and for some 2-in dia. holes, the primary blasting agent is a mixture of 77 per cent ammonium nitrate, 18 per cent molasses and 5 per cent water by weight. Between 5 to 10 per cent of the total charge is AN60 gelignite for priming and boosting.

A considerable amount of secondary reduction is necessary and during the three daily shifts many pops and plasters are fired using fast igniter cord and capped fuses; AN60 gelignite is used for pop holes and semigel for plasters.

POWER SHOVELS

In the early years of the opencut $\frac{5}{8}$-yd^3 steam shovels were used, the ore being transported by steam train. In 1937 two 1-yd^3 steam shovels and one 1-yd^3 diesel shovel were purchased, together with one 3-ft 6-in gauge steam locomotive and three $5\frac{1}{2}$-yd^3 side-tipping trucks. Two $2\frac{1}{2}$-ton electric locos were built from available plant.

Following the adoption in 1938 of the 7 000 000-ton scheme, an extensive mine preparation programme was drawn up. This included:

(i) the continuation of the unwatering of the mine and the reconditioning of the Main and the Linda shaft to the 750-ft level;

(ii) the installation of a 42-in by 36-in jaw crusher with ore pocket connections to the Main shaft and ore loading and transporting facilities at the 650-ft level; and

(iii) extensive tunnelling and winzing at selected levels to provide for transportation of ore and waste to transfer passes.

New equipment purchased to carry out this programme included two $1\frac{1}{4}$-yd^3 diesel shovels, one steam locomotive, ten 5-yd^3 side-dump cars, and twelve 2-yd^3 side-dump cars (26-in gauge).

At the end of 1944 two $2\frac{1}{2}$-yd^3 electric shovels were ordered and the introduction of more efficient transport was being investigated. One $2\frac{1}{2}$-yd^3 shovel arrived in 1945 and the second in 1946. Four owner-driven 12-ton petrol trucks began transporting from these shovels in 1946 and bulldozers became available.

In 1948 the 750-ft level crusher station was brought into operation. In the following year five $1\frac{1}{2}$-yd^3 shovels in almost new condition were purchased from U.S. disposals in Manila. The purchase of these shovels coincided with the start of a decade of rapid expansion which began with the opening of No. 7 bench from the 750-ft level in 1950, followed by the installation of a crusher at the 850-ft level in 1954 and the consequent opening up of No. 8 bench from this level in 1957.

Since 1950 the following shovelling equipment has been purchased: one new 5-yd^3 shovel, one second-hand $3\frac{1}{2}$-yd^3 shovel, two new $2\frac{1}{2}$-yd^3 shovels, three second-hand $2\frac{1}{2}$-yd^3 shovels, one second-hand $1\frac{1}{2}$-yd^3 shovel and two F.E. $1\frac{1}{2}$-yd^3 loaders.

Shovels at present available for use in the opencut, some of which are being rebuilt or overhauled, consist of:

one 5-yd^3 electric shovel
one $3\frac{1}{2}$-yd^3 electric shovel
five $2\frac{1}{2}$-yd^3 electric shovel
one $2\frac{1}{2}$-yd^3 diesel shovel
five $1\frac{1}{2}$-yd^3 electric shovel
one $1\frac{1}{2}$-yd^3 diesel shovel
two $1\frac{1}{2}$-yd^3 F.E. loaders

In the present operation, mining up to 90 000 tons of material per week, seven shovels are worked per shift.

OPERATION

Production

Before the day and afternoon shifts the mine manager and his assistant decide what production is required for the shift and with the senior foremen of the upper and lower benches select the shovels to be worked. The night-shift foremen are instructed in writing in a log book and also by the afternoon-shift foremen. At the beginning of the shift the shift foremen of the upper benches (i.e. No. 5 bench upwards) direct the contractors' trucks available to the shovels to be worked on the benches accessible by road.

In allotting the trucks to the various shovels the foreman has to take into account whether the trucks are three-axle or two-axle, the performance of individual trucks and drivers, the performance of individual shovels and drivers, the bench and the material being shovelled and whether the truck has a tailboard or not. If carting downgrade is required he selects the trucks with covered tail shafts for this job, because the hydraulic brake lines are protected in the event of a broken unit. He must ensure that the 5-yd^3 shovel is supplied with a minimum of six trucks which must not have tailboards because of the size of material being handled.

A usual distribution of trucks is shown in Table II.

TABLE II

	Shovel capacity yd^3	*No. of trucks*	*Average tons/shift*
No. 6 bench upwards	5	6	1650
	$3\frac{1}{2}$	4	1080
	$2-2\frac{1}{2}$	8	1500
	$1-1\frac{1}{2}$	3	510
Nos. 7 and 8 bench	$1-2\frac{1}{2}$	2	750
	$1-1\frac{1}{2}$	2	510
		Total	6000 tons

To ensure maximum production the shift foremen, who are primarily engaged with loading and transporting, constantly supervise the operation, the main points being as follows:

(1) insistence on double side loading wherever possible;
(2) the initial correct distribution of trucks taking into account the factors already mentioned;
(3) the correct planning of greasing time on day shift to ensure no loss of production (no greasing is done at the start of day shift until the shovel position with regard to repairs has been ascertained);
(4) the quick redistribution of trucks to spare shovels as soon as a breakdown occurs;
(5) good piles of materials being worked by shovels and clean-up along benches left to bulldozers and an F.E. loader; and
(6) the supervision of shovel drivers to see that operating instructions are carried out correctly. These instructions have been drawn up by the mine staff, some being:
    (*a*) on no account must sprag devices be used as a substitute for a face brake to hold the shovel while working;
    (*b*) 'side swiping' must be avoided;
    (*c*) the excessive use of the rack to push the bucket hard into the heap is to be avoided;
    (*d*) shovels must be kept close to the rock pile to allow the machine to be worked in the strongest part of the rack arms and to avoid over-racking;
    (*e*) cable plugs are to be kept clear of the ground;
    (*f*) tracks must not be allowed to sink deep into the ground.

At the end of day shift the foremen have to ensure that the shovels are in a safe position before blasting and that the power cables are safe. Cables should not be left trailing in water or lying beneath a loose face.

Where necessary, $1\frac{1}{2}$-yd^3 shovels have been used on roads at grades of 1 in 9. The shovel travels on the incline with a D8 bulldozer connected to it by a heavy sling as a precautionary measure and also to give assistance when needed. The shovel tracks are run diagonally into the face at the shovelling point and then the driver prepares a working area which is less steep than the road grade. In these situations care has to be taken by truck drivers in spotting their trucks to ensure that they will not tip over when loaded. Only single side loading is possible.

In constructing new outlet roads $1\frac{1}{2}$-yd^3 shovels or the F.E. loader are used, the latter being preferred on the steep grades such as the 1 in $7\frac{1}{2}$ grade between No. 5 and No. 6 bench. Of the shovels, a $1\frac{1}{2}$-yd^3 diesel shovel is preferred as there is no trailing cable to congest further the already narrow working area. Where shovels are used the area immediately before the face is flattened for it to work on.

On very narrow benches the F.E. loader has proved invaluable, being easier for movement than the 1½-yd³ shovel. In travelling shovels larger than 1½-yd³ on steep inclines two anchor bulldozers are used. For such shovels rules have been laid down for the operators as follows:

(i) avoid spragging on uneven ground;
(ii) avoid, as far as possible, travelling 2½-yd³ shovels with the top sprag dogs in the lowered position. This practice could result in bent upper sprag shafts or the accidental disengaging of both travel clutches with consequent loss of control; and
(iii) wherever possible the shovel must be travelled and spragged in the forward direction, that is with the drive tumblers to the rear.

Performance

Table III gives the performance of shovels of different capacities used under varying conditions.

TABLE III

Shovel capacity yd³	*Working conditions*	*Tons/6-h shovelling*
5	Fine material	4500
	Well broken material	3000
	Poorly broken material or restricted area (single side loading)	1500
3½	Well broken	1600
	Poorly ,,	1000
2½	Well ,,	1300
	Poorly ,,	800
1½	Well ,,	750
	Poorly ,,	450
F.E. 1½	Well ,,	850
	Poorly ,,	450

In an 8-h shift productive time will be lost for erecting cable legs, greasing, travelling, crib, operating repairs, handling large rejects and standby when high rills and/or faces are on the move. Thus the actual shovelling time is about six hours. In Table IV the average shovel

TABLE IV

Shovel yd³	*Tons/h*	*Average h/shift*
5	275	5·6
3½	180	5·87
2½	125	5·54
1½	85	5·22
1½ diesel	106	5·57
1½ F.E.	100	6·6

performance figures for the first half of the year 1962–63 are given. During this half-year the average weekly total material mined was 81 000 tons.

Overall shovel availability, including shovels under overhaul, is 75 per cent. The availability of the 5-yd³ shovel and the 3½-yd³ shovel is 80 per cent.

Table V gives the cost of hire of shovels for the same period as Table IV. The operators' wages are not included in this hire charge.

TABLE V

Shovel yd³	*Hire cost (pence/ton) (Australian)* Budget	Actual
5	4·8	3·8
3½	6·0	4·0
2½	5·2	8·5
1½	14·4	8·25
F.E.1½	11·2	12·8

The figure for the 2½-yd³ shovel exceeds the budget because expenditure on the major overhaul of two 2½-yd³ shovels was being incurred as a direct charge against the operation of that fleet.

Shovel-drivers' wages are charged to benches and not to shovels. A target tonnage is set and a bonus is paid on tonnage above this figure. Table VI gives the tonnage bonus paid for shovels of different capacity on their respective benches and this can be compared with the transport contract rates given later (Table VIII).

TABLE VI

Shovel yd³	*Bench*	*Target tonnage* Ore	Waste	*Rate pence/ton*
5 electric	4A	1100	1100	0·8
3½ ,,	4	950	950	0·96
2½ ,,	5	400	400	2·0
2½ ,,	6	400	—	1·6
2½ ,,	6	—	400	2·0
2½ ,,	7	400	—	2·0
1½ ,,	7	400	—	2·0
1½ ,,	8	300	—	2·0
1½ diesel	3A, 4, 4A	—	400	1·6

Features

A. Comparison of 1½-yd³ F.E. loader and 1½-yd³ shovels.—The mobility of the F.E. loader fits it for miscellaneous work around the surface and for short widely scattered jobs, but it is not used for working under high faces and for loading heavy rough rock. On the other hand, the standard $1\frac{1}{2}$-yd^3 shovel is a comparatively immobile machine, but it is robust and is considered safer when operating under the high faces of the opencut. A particular disadvantage is the acid water conditions underfoot which are more severe for the loader than for the shovel because of its lower clearance and lighter design; it will operate in up to 12 in. of water, whereas the shovel can operate in 18 in. Comparative discharge heights are 9 ft 3 in. for the loader and 11 ft 4 in. for the shovel.

An important part of the shovel loading cost is bulldozer attendance, the cost of which amounts to about 40 per cent of the operating expenditure or 20 per cent of the total expenditure. Bulldozer attendance required for the F.E. loader is much lower than for standard shovels. This machine has been used to good effect in constructing new outlet roads and in opening up No. 8 bench from the 850-ft level. On opening-up work in the early stages and loading into a specially constructed short-wheelbase truck the rate of loading was 60 tons/h.

B. Lowering equipment over high faces.—Benches 7 and 8 have no access road; consequently shovels, trucks and bulldozers have to be lowered and/or removed to or from these benches when required. The operation of lowering a $1\frac{1}{2}$-yd^3 shovel (weight 45 tons), down a slope 1 in 1, between benches 100 ft apart, is carried out in the following manner (see Fig. 4).

Two $2\frac{1}{2}$-yd^3 shovels (weight each 80 tons) are used as anchors, one on No. 6 bench, the other on No. 7. Two blocks are attached to the back of the shovel to be lowered. To the main axle of each anchor shovel a $1\frac{1}{4}$-in diameter wire rope is attached. This rope is passed through the correct block on the $1\frac{1}{2}$-yd^3 shovel, then back through a block anchored to the front axle of the $2\frac{1}{2}$-yd^3 shovel. The free end is attached to a D8 bulldozer as shown in the sketch. The blocks are attached to the shovels by $1\frac{1}{2}$-in diameter wire rope slings (minimum safety factor for this work is 6:1). As can be seen, a change in direction is needed to keep one of the bulldozers running along the bench. A D8 bulldozer on No. 8 bench is connected to the front of the shovel by wire rope so that a pull can be exerted to direct the shovel.

Two riggers direct the lowering operation. The man in charge stands at *A*, the other rigger standing at *B*. *A* directs *B* and the bulldozers on No. 7 and No. 8 bench. *B* directs the bulldozer on No. 6 bench.

The shovel is driven until the point of balance is almost reached. Care is taken to see that the shovel is pointed in exactly the required direction. The driver then leaves the shovel and the cable is disconnected. The bulldozer on No. 8 bench then pulls the shovel over the edge with the other two bulldozers taking the strain. When the bulldozer on No. 7 bench reaches the point where the rope has to be slipped from

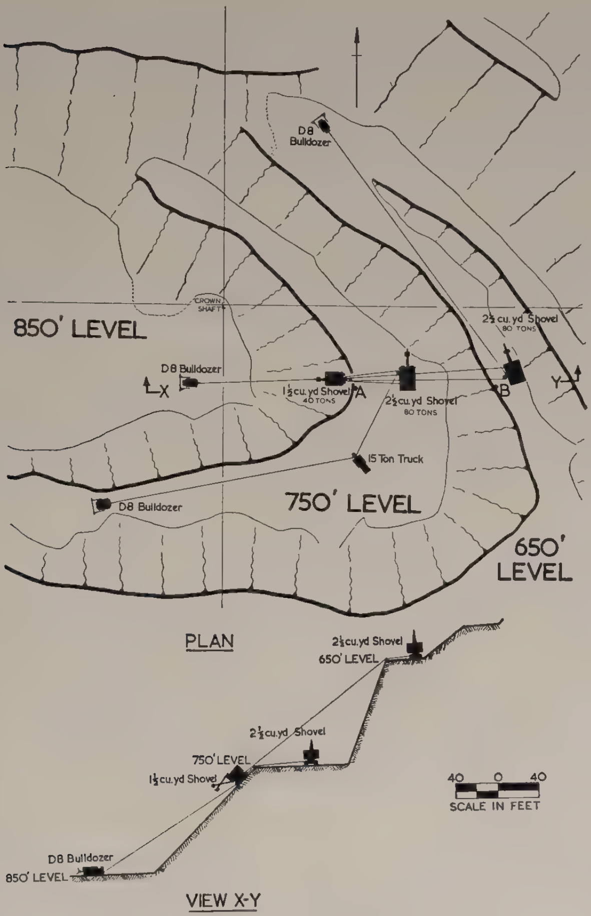


Fig. 4.—Method used to lower a $1\frac{1}{2}$-yd^3 shovel from No. 7 bench to No. 8 bench.

the block, the lowering operation ceases and the static load is taken by the other bulldozer. The actual lowering of the shovel takes $\frac{3}{4}$ h.

Maintenance

Maintenance of opencut machinery comes under the control of the mine mechanical engineer. The maintenance crew of the shovels consists of two foremen, three riggers, four fitters with four trades assistants, a boilermaker with an apprentice and one trades assistant, and two electricians with two apprentices.

The shovel crew inspect their shovel at the beginning and end of the shift; when a fault occurs it is reported immediately to the shift foreman, who then re-organizes his trucks if necessary and contacts the shovel maintenance foreman. If a fault occurs on either the afternoon or night shift the shift foreman has to decide whether calling in the shovel maintenance crew is warranted, depending on whether the shovel is in ore or waste, how large the fault is, the condition of the standing spare shovels and the availability of suitable material for shovelling. H.F. wireless communication and telephone are available in and around the opencut.

All maintenance jobs are carried out on the mine, either in the opencut or at the company's workshops. Major overhauls and rebuilds are carried out in the workshops, and damaged cables are repaired at the electrical workshop.

Overhauls

On the $1\frac{1}{2}$-yd^3 shovels it is estimated that £5000 has to be spent on overhaul for every 500 000 tons shovelled. On the $2\frac{1}{2}$-yd^3 shovels it is estimated that £5000 has to be spent on overhaul for every 1 000 000 tons shovelled.

The recent conversion of a $1\frac{1}{2}$-yd^3 shovel from electric to diesel, which included the replacement of some parts, cost £5100.

Two second-hand $2\frac{1}{2}$-yd^3 shovels purchased in 1959 were modified at the company's workshop to correct some defects encountered in similar models in use in the mine. The main axles were stripped out and two 2-in diameter shear dowels at 90° were put through the travelling clutches into the 9-in diameter main axle. Extra plating was added to stiffen the jib and other minor adjustments were carried out. The total cost for modifying these two shovels was £2546.

Rebuilds

When a shovel is to be rebuilt it is completely stripped. Many parts have to be replaced, the workshop jobs consisting mainly of manufacturing shafts, machining roughcast bushes and fabricating a new cabin.

The cost for $1\frac{1}{2}$-yd^3 shovels is up to £12 000 if the job can be done fairly quickly. The cost for $2\frac{1}{2}$-yd^3 shovels, including conversion to diesel electric, is up to £16 000.

BULLDOZERS

Bulldozers became available in 1946. The present fleet consists of:

No.	*Type*	*Use*
1	Caterpillar D6	Reserve ore stockpile
5	,, D8–8R series	Opencut
3	,, D8–2U ,,	,,
2	,, D8–H ,,	,,
2	Allis-Chalmers HD16	Engineering work
1	Super C Tournadozer	Road cleaning and pushing dirt

Operation

In the opencut these machines are used for cleaning up after firing, cutting toe, cleaning up around shovels, where necessary pushing material over to the next bench below, cleaning benches, maintaining roads, cleaning dumps, moving cable legs, towing wagon drills, pushing disabled trucks up inclines and acting as anchors for shovels being travelled down inclines.

The opencut bulldozers come under the control of the shift foremen and it is their responsibility to see that road repairs are carried out by these machines whenever they are spare. It is the responsibility of the day-shift foremen, at the end of the shift, to position bulldozers at the nearest safe point to the face so that roads can be opened up quickly for the start of the afternoon shift. Dumps are kept clean and consolidated by using a D8 bulldozer.

The overall cost of bulldozing and grading in the opencut and on the dump is 3·2d/ton.

TRANSPORT

In the early days of the opencut transportation was carried out by steam and electric trains. In 1946 four 12-ton payload, owner-driven trucks (petrol engines) were introduced to the opencut to cart material from the new $2\frac{1}{2}$-yd^3 shovels.

With the increase in the number of power shovels in 1948 it became necessary to introduce heavier trucks, with diesel engines, and 12 three-axle Diamond T 980 transporter chassis were purchased in that year from Army disposals. These tractor units were strengthened in chassis and springs and were fitted with 15-ton payload dump bodies and hydraulic hoists. The truck fleet consisted of this type of vehicle until 1955 when 15-ton payload two-axle trucks were introduced. Since then truck owners have gradually been replacing the Diamond Ts (which are no longer being manufactured) with two-axle trucks. These trucks carry approximately one ton more than the Diamond Ts. Truck loads weigh between 16 and 17 tons and may be as high as 20 tons.

The contractors' fleets consist of eighteen two-axle trucks and eleven Diamond T trucks. Mount Morgan Ltd. owns 31 Diamond Ts, of which four are operated on No. 7 bench and two on No. 8. Other trucks are hired out to the contractors if they require them.

Operation

Production

The distribution of trucks at the beginning of the shift has been mentioned in the section on shovels.

Certain rules have been drawn up for operating trucks and weighbridges, the more important being:

Safety regulations for truck drivers

1. There shall be no overtaking of vehicles on any bench within the opencut and three vehicle lengths clearance shall be kept.
2. Trucks bodies shall be down before leaving the dumps or breaker station.
3. Trucks shall change into the gear required to negotiate an incline, at the bottom of that incline.
4. No trucks shall be driven with faulty brakes, steering or lights.
5. No truck driver shall coast his truck in neutral.
6. No driver shall leave his truck while the engine is running.
7. Drivers must ensure that they are carrying a safe load.
8. No truck shall tip over any dump without a dump attendant being present.
9. Passengers may only ride in the cab or in the back when the truck is empty.

Rules for the safe removal of disabled trucks from the opencut

1. No disabled truck is to be moved before the shift foreman has inspected the truck and established the cause of the trouble.
2. The only approved method for moving disabled trucks on inclined roads is to push them out with a bulldozer. If a bulldozer cannot get behind the truck it can be towed to where the bulldozer can take over.
3. Trucks are to be taken out in daylight hours only.

Rules for 40-ton weighbridge operation

1. Each weighbridge must be manned by a checker when in use.
2. Shift bosses must watch empty trucks and are to weigh very dirty trucks. The tare of the truck thus weighed is then to be used throughout the shift.
3. A list of the heaviest tare recorded for each type of truck will be kept by the mine clerks and they shall use the applicable tare if the tare of a truck is not shown on the weighbridge card.
4. Trucks must stop before moving on to the weighbridge.

Wet weather restricts operations in the opencut and loaded trucks are not allowed to travel out from benches 4A, 5 and 6 because of the steepness of the grades.

At night in the opencut the drivers have found it more convenient for trucks approaching each other to switch off the lights.

As the opencut is classed as a mine, mining regulations have to be complied with. This means that dust has to be reduced to a minimum, and up to two watercarts are used continuously for the full round of shifts.

The company transport foreman acts as a liaison officer between the company and the contractors. He is responsible for checking shift reports on bulldozers and trucks and, with the mine manager, he fixes the contract price to be paid for the contractors' work. He is also responsible for all roads, especially those to dumps. Dump edges are kept clean and consolidated.

Regular inspections of trucks' brakes, steering and lights are carried out by the transport foreman and he writes up a report which is available for inspection by the mine manager and the inspector of mines.

Performance

The two-axle trucks carry slightly more than the Diamond Ts, but comparison of trip times would not be significant as the drivers usually stay in order.

Table VII gives a typical number of loads per truck shift for various shovels.

TABLE VII

	Shovel capacity yd³	*Tons/shift*	*No. of 16-ton loads/truck shift*	*Total loads*
No. 6 bench upwards	5	1650	17	102
	3½	1080	17	68
	2½	750	11 to 12	46
	1½	510	10 to 11	31
Nos. 7 & 8 bench	2½	750	23	46
	1½	510	16	32

As can be seen from the tons/shift these figures are for the lower and more general shovel performances.

The contract price for carting material is calculated with regard to a number of factors, which include the shovel, the bench, and the destination. Table VIII gives a number of examples of tasks and contract price. The hire on company trucks is approximately 50 per cent of the contract price.

TABLE VIII

Haul	*Lift* ft	*Length* ft	*Contract price/ton* s. d.
No. 3A bench to perimeter road	185	4900	1
,, 4 ,, ,, ,, ,,	250	5300	1 4·8
,, 4A ,, ,, ,, ,,	290	6100	1 8
,, 5 ,, ,, ,, ,,	380	8100	1 10·5
,, 6 ,, ,, ,, ,,	470	8700	2 5·5
,, 3A ,, ,, No. 1 mill road	20	3800	1 —
,, 4 ,, ,, ,, 1 ,, ,,	90	4200	1 3·2
,, 4A ,, ,, ,, 1 ,, ,,	130	5000	1 6·4
,, 5 ,, ,, ,, 1 ,, ,,	220	7000	1 7·2
,, 6 ,, ,, ,, 1 ,, ,,	306	7600	2 3·2
,, 4A ,, ore to No. 2 mill	140	5400	1 8
,, 5 ,, ,, ,, ,, 2 ,,	230	7500	1 10·5
,, 5 ,, to reserve pocket	0	1200	– 11·2
,, 6 ,, ,, No. 3A pocket	0	800	– 11·2
No. 2 mill bin to No. 1 mill stockpile	50	4000	1 7·2

Drivers are paid wages plus a bonus for loads above the minimum set for each carting task.

Overall cost of transport in the opencut is approximately 1s. 5·3d. per ton.

Overall availability of trucks, including spares, is 70 per cent and individual availability is 90 per cent or better.

Operating costs

Operating costs have been taken from figures prepared in 1959 when a review of haulage contractors' rates was carried out. Since then truck operators have become more cost-conscious so the figures, despite an increasing price trend, continue fairly stable. The figures in Table IX have been prepared for the operation of Diamond T trucks.

TABLE IX

Item	*% of cost*	*Remarks*
Fuels, oils, greases	16	Cost of fuel has fallen from 2s. 1d. to 1s. 6d.
Tyres and tubes	13·65	
Wages	36·4	Wages have risen from £14 11s. to £16 16s. a week
Spare parts and accessories	20·35	
Mount Morgan Ltd. services, general expenses and depreciation	13·60	
Total	100·00	
Cost/ton 1959	14·95d.	
Cost/ton 1963	16d.	

For the two-axle trucks tyres and spare parts are more expensive, but this increase in cost is counteracted by the fact that the two-axle truck has rather less need of maintenance and replacement of parts and also because the two-axle truck carries 1 ton more than the Diamond T.

Gross income per truck shift is about £16.

An analysis of spares used for Diamond T trucks in one year showed the following distribution of cost:

Item	%
Engine parts	45
Bodies and hoists	21
Chassis	10
Transmission	6
Brakes	4
Sundries	14
Total	100

Tyre performance

At Mount Morgan driving tyre life has been measured by the number of tons carried by a set of four rear tyres. This is considered reasonable because truck tasks vary considerably over the life of a tyre.

Although the figures given in Table X are not conclusive, only one set of four tyres for each type having been tested, they do give an indication of the quality of the respective tyres.

TABLE X.—*Life of 1400 × 24 × 20-ply tyres fitted to two-axle trucks*

Make	*Tons per set*	*Cost/ton* d.	*Remarks*
U.S. 1	100 000	1·94	
U.S. 2	85 000	1·96	
U.S. 3	60 000	2·7	
U.S. 4	66 000	2·8	At present rock lugs are being tested
Aust. 1	54 000	2·9	
Aust. 2	43 000	3·65	
German	48 000	3·65	
Belgian	36 000	3·85	
French	28 000	6·02	These tyres would probably perform better where road conditions are perfect
Japanese	—	—	This make of tyre gives excellent service in Diamond T trucks

Notes

(1) A four-wheeled rock buggy type vehicle, with separate prime mover, was tested in comparison with the Diamond T truck. The tests were carried out in 1956 over a period of five months. These machines performed quite well, but were not purchased as a replacement mainly because of their inability to climb steep wet grades without wheel spin which resulted in hazardous driving conditions and heavy wear of the front driving tyres. Traction was improved by adding 1000 lb of water at 50 lb/in^2 to each front tyre.

Comparative figures taken over four months using the same route for both types are:

	Tons/load	*Loads/shift*	*Tons/shift*
Rock buggy type vehicle	17·65	14·8	262
Diamond T truck	16·1	14·4	231

The net load of the rock buggy type vehicle is 10 per cent higher, with cycle time 5 per cent shorter and shift capacity 13 per cent higher. The operating costs, including usage, tyre, fuel and lubricants, could be up to £4 8s. higher per shift than relevant Diamond T costs, while maintenance costs would be £2 8s. less per shift than the Diamond T costs, bearing in mind both running maintenance and periodic overhauls.

(2) Side-dumping trailers for use with Diamond T prime movers were considered as a hauling unit, but were rejected because (*a*) too large a turning radius is needed; (*b*) the coupling between the prime mover and the trailer is a weak link in a hard-rock opencut; (*c*) 18 tyres would be required compared with 10 in the Diamond T; (*d*) waste could not be dumped over the side of the dump and a longer length of dump would be required; (*e*) if the dump began to move there would be no chance of driving the truck out of trouble.

(3) Diamond T trucks are used underground on the lower benches to tip into crushers and pockets. Regular tests of exhaust gases are carried out to check for the following limits: 1 part per 1000 of CO in exhaust, and 1 part per 10 000 of CO in drives.

DISPOSAL AND UTILIZATION OF WASTE

The selection of dumping areas which will give the lowest possible costs has always been a problem because of the limited number of suitable areas available close to the opencut.

Before 1958 there were four main dumping areas, but since then, after a report by the mine superintendent setting out the comparative dump capacities, haul distances, lifts and estimated haulage costs, the Airstrip dump, the Perimeter dump and No. 1 Mill dump have been used. The maximum grade of all dump roads is 1 in 20 to allow top gear haulage.

A No. 12 grader is used to keep opencut and dump roads in good condition and a D8 bulldozer keeps the dumping area in good condition. One dumpman is stationed at each tipping point to direct the truck drivers when reversing to the tipping edge.

Selected waste is used for the construction of tailings dams, such as the Sandstone Gully tailings dam, anti-pollution dams, drainage control works, roads, railway embankments and a category 3 airfield 2200 ft long and 200 ft wide.

Crushed rock is sold for railway ballast, crushed felsite is used for road metal, and crushed porphyrite dyke is used for bitumen screenings and concrete aggregate. Crushed siliceous rock (85 per cent silica), is used for flux in the smelter.

ADVANCE OVERBURDEN REMOVAL

Because the Sugarloaf orebody is deep-seated, overburden in excess of the average overburden to ore ratio has been removed. This is termed 'advance overburden' and the cost of its removal was debited to an Advance Overburden Removal Account. This account will be redeemed when the overburden to ore ratio falls below the average ratio.

LOADING AND TRANSPORT SHIFT RECORDS

Weighbridge cards for each truck, and the shift foreman's report on shovels and trucks, are checked by clerks, who correlate them on daily sheets for trucking and shovelling. Shovel tonnages, some of which are used for calculating the bonus payable to drillers and powder monkeys in particular working areas, are totalled fortnightly to calculate pay, and monthly for the period reports. The tasks and tonnages carried by contractors' trucks are recorded so that the transport foreman can calculate the payments due to the contractors.

Acknowledgement.—The author wishes to thank the board of directors of Mount Morgan Ltd. for making data available for inclusion in this paper.

622.271.3:622.341.1 (477) *Paper 15*

Opencast Mining of Taconites in Krivoi Rog, Ukraine

M. G. NOVOZHILOV, Dr. Sci. (Techn.)

Professor, Chair of Opencasting, Institute of Mines, Dnepropetrovsk, USSR

SYNOPSIS

There has been widespread growth in the opencast mining of ferruginous quartzites in the Krivoi Rog iron ore basin, followed by concentration. The paper deals with the mining and geological features of the deposits, the hardness of rock, drilling and blasting methods, excavation of overburden and loading of broken material, transport by rail and motor, and piling of overburden. The aims of further research are stated.

The Krivoi Rog iron ore basin is one of the largest in the Soviet Union. Iron ore is mined both by opencast and underground methods. Ferruginous quartzites are mined by the opencast method.

There are two basic types of iron ore deposits from the point of view of their specification for opencast mining:

(1) deposits of ferruginous quartzites represented by intensely compressed synclinal folds: mining of such deposits is characterized by the need for comparatively small amounts of capital equipment and development work and a low stripping ratio (0·1 to 0·2 m^3/t) (metric tons);

(2) quartzite deposits represented by thick (250–300 m) beds of moderate and steep dip: mining of these deposits is connected with considerably greater amounts of development work, the average stripping ratio reaching 0·5 to 0·6 m^3/t.

By the nature of the ore minerals the ferruginous quartzites are subdivided into two main types:

(*a*) magnetite (non-oxidized) ores easily concentrated by magnetic separation; and

(*b*) hematite–martite (oxidized) ores difficult to concentrate by magnetic separation. Oxidized ores occur in a contact zone with magnetite ores and are mined incidentally.

TECHNOLOGY OF OPENCAST MINING OF FERRUGINOUS QUARTZITES

Stripping and preparation of deposits for systematic mining is planned as follows.

A site for the quarry railway station is selected with regard to convenience of serving the quarry and the industrial ground on which the concentrating mill is situated. Stripping is done by means of outer cuts which within the limits of the quarry contour lead to entrance cuts baring one or several mining levels with the aid of working cuts.

Inclines of outer and entrance cuts are assumed as 2·5 to 3·0 per cent depending on the quarry dimensions and the expedience of reducing the number of tight-end rounds.

The height of benches on upper levels consisting of loose and mixed rocks is 10 m, whereas the height of benches on lower hardrock levels is 15 m. The angle of slope of the upper benches is 45° and that of the lower slopes is 70°. The general angles of slope of the quarry sides varies from 25° to 40°.

Quarries, crushing and concentrating mills work three shifts per day, the estimated number of working days being 300 per year.

Drilling and blasting work

Blast-holes are drilled by means of heavy churn or well drills with rods weighing 2·5–2·7 t. The bits used are chisel-shaped with the edge 210–260 mm long. When drilling rocks with hardness according to Professor M. M. Protodyakonov's scale $f = 12$–18, the drilling rate varies from 5·4 to 7·4 m per shift (7 hours).

Recent improvements in drilling performance have been achieved by the use of jet-flame cutting and rotary machines using roller bits.

In 1962 at one of the quarries over 28 000 m of blast-holes were drilled with the aid of jet-flame drills, the hole diameter being 180 mm, with an average drill productivity per shift of 18·5 m.

In 1962 the problem of increasing the diameter of the lower part of the hole up to 300–350 mm was successfully solved by jet-flame drilling, which considerably increased the efficiency of blasting work. At present at this quarry seven jet-flame drills are successfully employed for routine production. A typical feature of jet-flame drills is the fact that their productivity increases with increase in the rock hardness.

Rotary drills are effectively used for drilling rocks with hardness f from 6 to 14 and jet-flame drills for drilling rocks with hardness f over 14 according to the Protodyakonov scale.

Blasting work

The blasting method was introduced for mining ferruginous quartzites during the period 1954–57. Bore-holes of 220–240 mm diameter were

TABLE I

Quarry no.	1	2	3	4	5
Rock hardness according to Professor M.M. Protodyakonov's scale	6 to 20	6 to 18	6 to 16	6 to 14	8 to 16
Bench height, m	10 to 15	10 to 15	12 to 40	12 to 28	5 to 15
Distance between holes, m	7 to 10	7 to 13	8 to 8·5	8 to 10	7 to 8
Toe burden, m	7·5 to 12	11 to 12	9 to 9·5	8 to 9	7 to 9
Charge concentration factor	1·0 to 0·9	1·0 to 0·9	1·0 to 0·9	1·1 to 1·0	1·0 to 0·9
Specific consumption of explosives, kg/m^3	0·39 to 0·87	0·64 to 0·60	0·58 to 0·60	0·35 to 0·45	0·56
Rock blasted per metre of bore-hole, m^3	59·1	54·0	55·3	86·7	52·4
Large rocks, per cent	up to 1·0	0·5 to 1·2	up to 1·0	0	0·5 to 1·0

drilled in one line and blasted in groups, each group numbering from 5 to 40 bore-holes. Depending on the properties of rocks blasted toe burden on the bench foot amounted to 9–12 m, the distance between bore-holes being 4·5 m. Owing to the small number of bore-holes the fragmentation was insufficient. The amount of rock blasted did not exceed 36 m^3 per metre of bore-hole, the specific consumption of explosives being 0·40 kg/m^3.

The use of drill bits of 260-mm diameter permitted an increase in the distance between bore-holes, which was also favoured by the use of multiple-row delay-action blasting. As a result, 36 m^3 of rock blasted per metre of bore-hole in 1956 increased to 86·7 m^3 in 1962.

In succeeding years, as the method of delay-action blasting was perfected, the production indices of blasting work were considerably improved. Details of results at a number of sites are given in Table I.

To improve the quality of blasts within recent years the working of high benches has been developed.

At the Krivoi Rog quarries high benches are blasted in rocks with thin layers and with hardness, *f*, of 6–14. It is usual for from 4 to 10 rows of bore-holes to be blasted before the rock broken by the previous blast is removed. This favours fuller use of the blast energy, which permits an increase in the distance between bore-holes and reduces the specific consumption of explosives.

At present the parameters of the bore-hole distribution for blasting high benches are the same as for ordinary benches (Fig. 1). The degree of fragmentation of rocks and ores is good.

From 1960 to 1963 at the Krivoi Rog quarries about 11 000 000 m^3 of rock was blasted using the method of high benches. The complete absence of large pieces and exceptional uniformity in fragmentation are typical of the rock blasted. This may help in solving the problem of applying a system of blasting which allows continuous loading from several benches in the opencast mining of hard rocks.

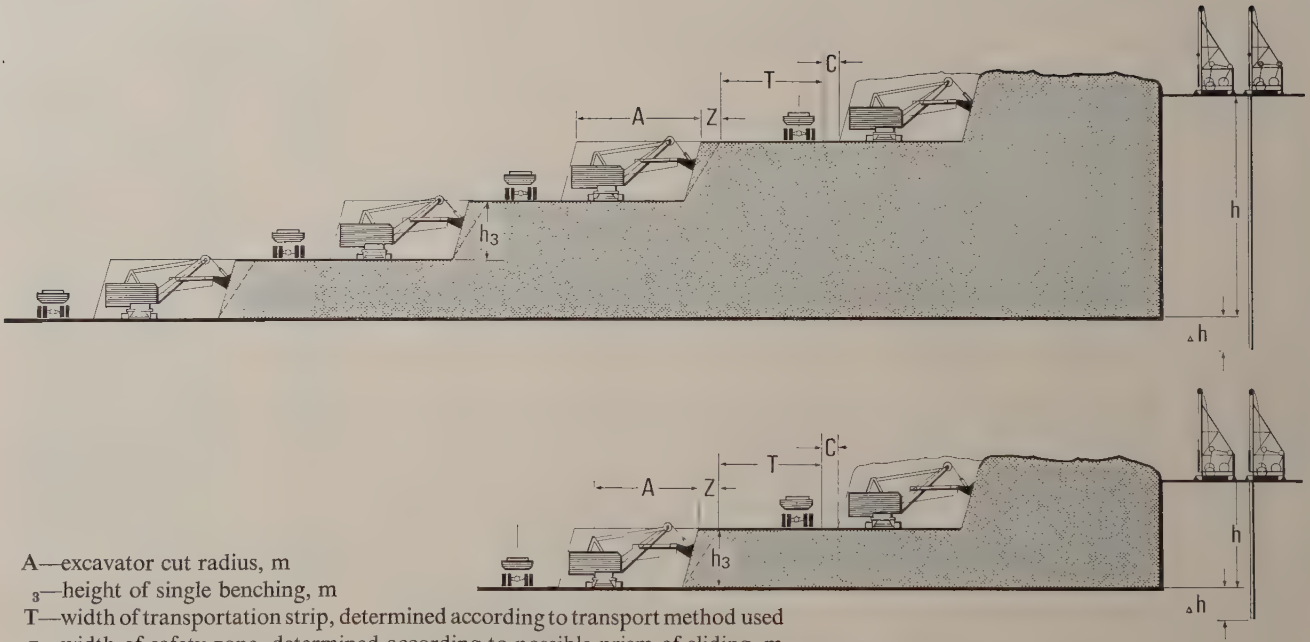
Excavation

Excavation of soft overburden and loading of blasted rock for transport are done by single-bucket excavators of types СЭ–3, ЭКГ–4 and ЭКГ–8 with bucket capacities of 3, 4 and 6 m^3 respectively.

As a rule, hard rock and ferruginous quartzites are loaded by excavators with a bucket capacity of 3 m^3. Overburden rock is extracted by 4- and 6-m^3 bucket excavators.

The utilization of excavators reaches 88 per cent so that only 12 per cent are in reserve or under repair.

The average annual output of one excavator amounts to 1 600 000 t, with an average specific weight of the material excavated at 2·7 t/m^3;



A—excavator cut radius, m
$_3$—height of single benching, m
T—width of transportation strip, determined according to transport method used
z—width of safety zone, determined according to possible prism of sliding, m
C—gap between low edge and transportation strip, m
h—height of multiple bench, m
Δh—depth, m

Fig. 1.—High bench mining method.

76·7 per cent of the excavator material is composed of hard overburden rock and ore.

Quarry transport methods

At present both railway and road transport are used at quarries. In 1961, of the total volume of 27 700 000 m^3 of rock about 17 000 000 m^3 were carried by road transport.

Railroad rolling stock at quarries consists of 80-t (and less frequently 60-t) dump cars, electric locomotives ranging from 100 to 150 t and heavy two-section diesel locomotives of 252 t. The average annual productivity of a train hauled by an electric locomotive reaches 1 300 000–1 500 000 t, where the average haulage distance is 5·4 km and the incline is up to 3·5 to 4·0 per cent of the distance to be covered.

The net train weight for a section of diesel locomotive is 480 t, and its annual productivity is 900 000 t or 9 000 000 t-km.

10- to 25-t locally built dump trucks are used for transport within quarries.

When developing a quarry, motor transport is used to a depth of 80 m, after which there is a combination of motor and conveyor transport.

Piling of overburden

Depending on the transport available, the excavator or bulldozer method of piling is adopted for quarries.

Excavators are employed with rail and bulldozers with motor transport.

The output of excavators in piling is higher than in a face by 35 to 50 per cent and exceeds 2 000 000 t per year. 100-, 140- and 250-hp bulldozers are used, the annual productivity of 140-hp bulldozers totalling 600 000 to 750 000 t, and that of 250-hp bulldozers amounting to 1 000 000 to 1 200 000 t.

It should be pointed out that the manpower productivity for bulldozer piling is five to six times higher and the cost two to three times lower than those in excavator piling.

FURTHER RESEARCH IN THE TECHNOLOGY OF OPENCAST MINING IN KRIVOI ROG

The principal aims of scientific research are the further study of characteristics of the deposits mined, methods of mining, efficiency of equipment, methods for improving technical and economic indices of ore mining as well as labour conditions of mine workers.

A wider application of jet-flame and rotary drills should considerably increase labour productivity and reduce the cost of drilling holes, and work on this is being done.

Blasting of high benches in a dense medium providing for good and uniform crushing of hard rocks will, within the next few years, allow the use of rotary excavators for loading blasted rock on to belt conveyors, which will considerably increase mining efficiency,

Investigation in the field of transporting rock will have as its purpose the improvement of railway and motor track network as well as the development of more efficient haulage equipment.

Studies for perfecting the methods of concentrating iron ores, and in particular non-oxidized iron ores, will be continued.

With the expansion in the scale of iron ore mining in the USSR, in order to meet the requirements of the national economy even greater importance is attached to scientific research.

Discussion

(Papers 13–15)

Mr. T. M. Dover,* introducing Papers 13 to 15, said that Dr. Flachsenberg's paper, 'Loading and haulage in quarries', described those operations in a large limestone quarry with an output of 10 000 to 15 000 t per day. As a practical engineer the author had taken as a basis for the design of a quarry the size of the crushing plant. From that he had determined the maximum size of bucket on the loading shovels as a means of limiting the maximum lump size acceptable in the crusher. That was a concept of planning fairly widely used, but possibly a discussion on its validity and on whether other forms of lump size control could be substituted would be of value.

In a similar way the author planned the maximum size of haulage vehicle body by the amount of material that could safely be dumped at one time into the crusher feed. Discussion on feeding of crushing plant might result from that view.

The ideal situation of the crusher at the loading point was mentioned and perhaps the author would give his views on a fully mobile crushing plant on crawler or other mountings so that it could follow the loading machine, thereby eliminating transport of run-of-mine rock.

An interesting comparison of the proportions of total cost due to winning, crushing, loading and haulage was given, and from that it might be possible to make comparisons with similar operations in other countries with different monetary systems, if the cost of one of the operations were known.

Investigations into improvements in blasting methods were mentioned and he felt sure questions on the latest developments would be asked, particularly as the problem of oversize in blasting was always present. Although that was a subject outside the scope of the paper, he was certain it would be of interest.

The method of calculating the efficiency of 10-, 20-, 30- and 40-t capacity trucks and the advantage of double and single spotting at the loading machine and the number of trucks required for hauling up to 600 t/h over distances below 1000 m would be useful as a guide to those who had to design similar systems in the future. The consideration of loading and haulage costs showed how careful operators must be in selecting the right size of equipment; it was not a job amateurs should undertake lightly.

---

*Ore Mining Branch of The United Steel Cos., Ltd., Lincolnshire.

In connection with bucket design of loading machine, the author mentioned that with large-diameter bore-holes used for blasting the rock pile was more uniform and not so tightly packed. That was probably a factor in blasting that was often overlooked, especially with stratified rock. The interlocking between fragments must be broken to give good loading conditions.

The pros and cons of building up wearing parts, particularly excavator teeth by welding, were mentioned, and that was a matter which could lead to an interesting discussion. He hoped that engineers responsible for excavator design would take some part as interchange of ideas between designers and users could not but be beneficial to both. In that connection he would draw designers' attention to the author's suggestion that there was a need for a continuous loader in quarrying. With that he believed all operators would agree. The reduction in cost in favour of rubber-tyred haulage compared with rail was interesting and it was hoped further particulars of the value of both systems would be brought out in the discussion.

In 'Loading and transport: experience at Mount Morgan, Queensland', Mr. Sheil gave an outline of the historical development of that mine, started in 1882, and as an underground mine with a change to opencast operations beginning in 1932. The scale of the operations was shown by the fact that over 1 500 000 oz of gold and 136 000 t of copper had been produced from 22 000 000 t of ore, requiring the removal of nearly 46 000 000 t of overburden by 1962. From that it could be seen that an ore : overburden ratio slightly worse than 1 : 2 had to be faced. Perhaps the author might like to indicate whether the ratio of ore to overburden in future would increase or decrease. Was he also correct in assuming that the greater part of the roof material of the deposit had now been removed and that future operations would only require the removal of that part of the overburden overlying the sides and barren rock necessary to give safe slopes?

Drilling and blasting appeared to be extremely difficult in an area of varying rock strength and an interesting discussion on the advantages of ammonium nitrate molasses slurry should result, and also on how that slurry was mixed and handled, and what advantages it had over the more common AN–FO mixes.

Perhaps, also, the author could give an indication of drilling patterns and the method of loading holes under various conditions.

The problems arising from the use of underground workings to carry the transport system to an opencast mine were also of interest, and the method of feeding underground crushers from rock dug with a 5-yd^3

shovel could be compared with Dr. Flachsenberg's suggestion of matching crusher and face shovel bucket size.

The system of using owner-driver contractor trucks was unusual in the United Kingdom and a discussion on the organization necessary to utilize owner-driven vehicles might result.

The third paper to be discussed was 'Opencast mining of taconites at Krivoi Rog, Ukraine', by Professor M. G. Novozhilov. They particularly welcomed papers from the USSR and hoped the author and his colleagues would be encouraged by the reception of the paper to send more for publication in the United Kingdom. They hoped also that some of their methods might be useful in the USSR, and that they might learn of new techniques from the present paper.

In the deposit described the extremely hard ferruginous quartzites were overlain by moderate overburden, the ratio of ore to overburden being 1 : 6 m^3 per metric ton. They were very interested in the layout of the pits and he hoped it would be possible for the author to provide diagrams showing the plan view of typical pits.

On the question of drilling, the change from heavy churn drills to jet-flame cutting and rotary drilling with roller bits was in the same direction as development in the West. It was hoped that some of the members who used these techniques would take part in the discussion and give their own experience with those methods. The system of drilling and blasting a number of benches at the same time was unusual and it would be interesting to know of the advantages and disadvantages of the method compared with the more usual use of drilling single-bench depth at a time.

The author mentioned the utilization of excavators and referred to 88 per cent availability and only 12 per cent in reserve or under repair. The method of calculation of utilization varied considerably in different parts of the world and in different companies in the same country. Discussion on that might lead to the adoption of a common method of calculation.

Similarly, outputs of excavators were very dependent on bucket size, so that when considering annual outputs one would like to know the size of bucket being considered. On quarry transport methods the size of railway wagons varied considerably and was dependent on so many factors that that interesting subject could easily form the basis of a paper on its own.

Interesting figures of annual productivity were given and there again the advantages of statistics based on annual figures, as against monthly, weekly or daily ones, might be considered.

Regarding the use of conveyor for the deeper workings, perhaps they could have some information on the methods of transferring from motor transport to conveyors, and whether at that stage the ore was crushed.

The author presented some very interesting glimpses of the research which was being done, and the possibility of blasting hard rock sufficiently for rotary excavators was put forward. The concentration of iron ore was, of course, a matter that was outside the scope of the symposium, but one of paramount importance to the industry throughout the world.

Finally, he would suggest that among the main points for discussion might be:

(*a*) The situation of the crushing plant relative to the loading point;
(*b*) The possibility of improving blasting methods and materials to give better displacement and fragmentation;
(*c*) Whether the use of deep holes to blast several benches at a time was applicable generally;
(*d*) The importance of correct relative size of loading machine, and also of equipment and crushers;
(*e*) The advantages of a continuous loading machine as against a single bucket excavator for hard rock; and
(*f*) The possibility of standardizing utilization or productivity figures.

Mr. T. T. Heywood* said that Dr. Flachsenberg referred to the dominating position that the size of a crusher occupied in relation to loading, hauling and winning of quarried material. He also emphasized that the shovel bucket should be dimensioned so that it could handle pieces of rock *only* up to the largest size that could conveniently be passed through the crusher. The question of crusher size was one which could be emphasized even more strongly. The life of a crusher was much longer than the life of loading equipment and when planning a quarry layout one had to discount any manufacturer's references to crusher capacities, which should be sized not to what was required at the outset of an operation but to take the feed which would be produced in the future when increased production was being obtained with the larger equipment which would undoubtedly be used in times to come.

Mr. T. F. Barclay† asked Professor Novozhilov if the two-section diesel locomotives could operate as separate units and whether the net train weight when close-coupled was 960 t. He also wondered if the electric locomotives were trolley or battery locos, and said he would welcome details of the comparative operating and maintenance costs of the electric and diesel locos.

---

*N. V. Mijn Maatschappij, Curaçac.
†Stewart & Lloyds Minerals, Ltd., Melton Mowbray.

Mr. S. C. Brealey* said that he would like to comment upon Professor Novozhilov's reference to possible future developments in continuous loading machines operating in blasted ground. Manufacturers of those machines had avoided that problem in the past, but he was convinced that the difficulties involved would be overcome in the not too distant future.

The continuous machines, such as the bucket-wheel excavator, avoided the mechanical and electrical peak loadings which occurred in the case of the shovel and the dragline. That made the machine more economic and more efficient—where it could be applied. The digging action was difficult—the shovel had a crowd action, whereas the bucket-wheel excavator scraped the ground like a milling cutter. Therefore if the bucket-wheel excavator principle were to be used, the ground must be well fragmented and to maintain a satisfactory working face would probably entail buffer-blasting. It would, however, appear to be more difficult to develop a machine using a rotary action with relatively large rock buckets or a reciprocating action with one bucket which could deliver directly to a continuous conveyor system. If lump size were to be matched to the bucket then a crusher incorporated in the machine could be used to reduce that lump size to one suitable for the conveyors. Bearing in mind that machines of some 7000-t service weight could be mounted on crawlers that would appear possible. Indeed, crawler-mounted crushers had been built for similar purposes.

Mr. R. J. B. Kenna† said that he agreed with Mr. Dover that there was a need for standardizing such terms as 'shovel availability'. Before that term could be fully appreciated it was necessary to know the period of time over which the machine was expected to work. Could Mr. Sheil expand his statement relating to shovel availability on page 333. He noted that Professor Novozhilov referred to 88 per cent utilization of excavators and felt that figure required further explanation if it were to be properly appreciated.

In the section on maintenance Mr. Sheil wrote (pp. 335–6): 'The shovel crew inspect their shovel at the beginning and end of the shift; when a fault occurs it is reported immediately to the shift foreman'. The author also mentioned major overhauls and rebuilds. Was there, in fact, a planned or preventive maintenance programme on either a weekly or monthly basis and, if not, was there any reason for that?

Professor Novozhilov stated that the choice between jet-flame drills and rotary drilling depended on rock hardness. Would he not agree that

---

*Powell Duffryn Technical Services, Ltd.

†Ore Mining Branch of The United Steel Cos., Ltd., Lincolnshire.

'spallability' (a factor dependent upon the coefficient of expansion of the minerals in the rock) and the degree of fracturing were also important factors when deciding upon the method of drilling?

For instance, at the iron ore operation at Wabush, Quebec, where the rocks were schistose, jet piercing was unsuccessful in rocks containing high proportions of the mineral anthophyllite. That was due to the poor spalling characteristics of the mineral, and jet-flame drilling thus became uneconomic. In fractured zones and on crests of banks where rock was broken rotary drilling was again preferred.

Perhaps the author would give details of the Protodyakonov scale of hardness; that would enable comparisons to be made with scales in use in the United Kingdom.

Mr. J. C. Hood* said that he had recently been employed on opencast iron ore mining in Sierra Leone, where all earthmoving equipment, including face shovels, was operated by Africans. One of their chief problems was the maintenance of plant—so long as the plant was moving, production was no problem. There was, however, considerable loss of time due to minor breakdowns in addition to down time for planned maintenance. In an effort to overcome that, the question of whether the operator should be responsible for the cleaning and servicing of his machine was raised. If so, he naturally took a keener interest and that should lead to a reduction of lost time. In fact, however, he was somewhat reluctant to help with the cleaning and was often insufficiently skilled to be made fully responsible for the servicing. The result was that he usually took no part in either, with detrimental results on plant availability.

He would appreciate Mr. Sheil's comments on the following:

(*a*) Did the operators of shovels, bulldozers, dumpers, etc., take any part in the cleaning and daily servicing of their machines?

(*b*) Assuming that the mine at Mount Morgan was run on a three-shift basis, how long was allowed for the routine daily servicing of shovels, and was any fixed time laid down for such servicing?

(*c*) What arrangements were made for the mobile servicing and fuelling of bulldozers, it obviously being uneconomic for that to be done at fixed points in the pit.

---

*Mining Engineer.

Mr. J. G. Lancaster* asked Dr. Flachsenberg if he would clarify the size of rubber-tyred shovel referred to in his paper, and also comment on the behaviour of such machines on sloping, wet, smooth ground when attempting to dig into solid piles.

Mr. J. C. Hood's comments on operators assisting with the servicing and maintenance of their own machines led him to relate the pattern carried out at his own Company's sites. Each operator was compelled at the commencement of each shift to check his machine's oil and water levels and to grease up at prescribed points. The major part of the daily servicing, however, was done from a mobile servicing and fuelling unit, during meal-breaks wherever possible, especially if it was an excavator serving lorries.

All servicing was done on the machine's run to eliminate wastage of time by withdrawing the plant to a maintenance area. Regular maintenance schedules demanded the withdrawal of a unit, but as those were determined in advance, deployment of all units could be programmed for greatest utilization.

Turning to another matter, he would welcome suggested methods of hoisting 500 t of mineral clay per day (eight-hour shift) from a quarry with vertical sides of 80–90 ft and loading into lorries at surface. To the present time inclined quarry roads had been in order, with ample space to provide a reasonable gradient for lorries but, because any quarry road would be cut through successive strata of wet sands, clayey sandstones and Fullers' Earth clay, and be of very great length, the upkeep and maintenance costs could well be excessive.

The hoisting unit would be fed by a rubber-tyred shovel, but at the same time would have to be mobile as a whole, as the quarry face would advance quite quickly.

WRITTEN CONTRIBUTIONS

Mr. A. Major-Stevenson:† I would like to comment on Dr. Flachsenberg's reference to the high cost of welding up shovel dipper teeth when they became worn. The author touched on the fact that efforts were being made to counteract tooth wear by means of 'fit-on' pieces. I would point out that a range of 'two-part' teeth is now widely available both in the United Kingdom and the U.S.A. They have tips, or points, which are quickly replaced when worn but, furthermore, they have self-sharpening characteristics so that their digging performance is

---

*The Fullers' Earth Union Ltd., Redhill, Surrey.
†Ruston-Bucyrus, Ltd., Lincoln.

retained throughout their life. That, of course, as with conventional one-piece teeth, varies according to the nature and abrasiveness of the material being loaded by the shovel.

Monsieur M. Beulaygue:* I shall be grateful if Dr. Flachsenberg can let me have the hourly costs of haulage for one 10-t, one 20-t and one 30-t truck, and those of a face shovel loading with a bucket of which the capacities are 1 m^3, 2 m^3 and 3 m^3 respectively.

---

*Commissariat à l'Energie Atomique, France.

Authors' Replies

(Papers 13–15)

Dr. P. Flachsenberg: In reply to the questions posed by Mr. J. G. Lancaster (p. 357), the rubber-tyred wheel loader has, as shown on page 319, with 150 horsepower, the same power as the crawler loader. The shovel has a capacity of 2·3 m^3.

The wheel loader has steel safety chains which ensure a firm stance on all types of ground so that the shovel can dig into solid piles even when operating on slight slopes.

The reply to the question by Monsieur M. Beulaygue (p. 358) regarding the operating costs per hour for trucks of different capacities cannot be given simply. Presentation of such details is not usual as there are a great number of determining factors involved. A characteristic value for the trucks is the cost per ton. Figs. 8 and 9 on pages 311 and 312 give information on this.

The rubber-tyred wheel loader had very high initial tyre costs. For the protection of the tyres, and to increase stability and grip, wire nets (tight chains) are drawn over them.

It will be of interest to learn that when used in opencast workings the life of a set of special tyres—without wire nets—of a 150-hp wheel loader was 1500 operating hours. A new set of tyres can accommodate two wire nets (chains) and the life of the tyre becomes approximately 7000 operating hours. The life of a new set of tyres is equivalent to one replacement of the wire nets (chains). We thus get the following specific tyre costs:

Tyre costs per operating hour *without* wire chains
= 9.15 DM/operating hour;

Tyre costs per operating hour *with* wire chains
= 5.05 DM/operating hour.

Mr. G. Sheil: In answer to Mr. T. M. Dover (p. 352), the greater part of the overburden has been removed and the majority remaining is made up of side burden and waste within the limits of the ore reserves. The ratio of ore:overburden is at present 1:2, remaining at this ratio for two years, and will then decrease over the remaining life of the mine.

As Mr. R. J. B. Kenna states (p. 355) the term 'shovel availability' should be qualified and on p. 333 the term should be 'shovel mechanical availability' and is based on an 8-h shift. The 25 per cent mechanical down time for the shovel fleet includes mechanical and electrical repairs and major overhauls.

The planned shovel maintenance programme at the opencut is not based on tons moved or hours worked, as is the case with bulldozers. The mechanical foremen make daily inspections of the shovels and note defects which will shortly require attention. The repairs required are then programmed to suit both the production and maintenance plans. Major overhauls are planned on the mechanical condition of the shovel which is assessed by experienced mechanical staff. Maintenance of the engines of the diesel shovels is the only maintenance which is programmed on an hourly basis.

To answer Mr. J. C. Hood's question (p. 356) on shovel and bulldozer servicing and fuelling, a description of these operations as practised on the three-shift operation at Mount Morgan is as follows: A mobile grease truck equipped with two air-operated grease guns, oils, gear compound, kerosene and waste is manned by a shovel operator and a truck driver, both of whom do the shovel greasing. The operators on the shovel are required to assist in the greasing and it is their responsibility to service the boom, rack arm and ropes, as well as keeping their shovel clean.

This servicing operation is carried out on day shift and takes between 20 and 30 min a shovel to complete. The order in which the shovels are to be greased is arranged each morning between the mechanical foreman and the operating foreman to fit in best with production requirements.

On afternoon and night shifts the shovel operators are responsible for greasing the important sections of the track equipment and the high-speed portions of the shovel machinery. Fifteen minutes a shift are allowed for these operations.

For fuelling and servicing bulldozers a 3-t truck is equipped with a 500-gal tank of diesel fuel, a petrol-driven fuel pump and an accurate flow-meter. The truck is also equipped with air cleaner and transmission oils and hand-operated grease guns.

The truck driver is responsible for fuelling the machines and for noting fuel consumption and hour-meter reading. The bulldozer driver is responsible for greasing, oil checking and oil cleaner servicing, as well as the general cleanliness of his machine.

GROUP III—MISCELLANEOUS

Session 5—Computation (Paper 16)
Soil Mechanics (Paper 17)
Groundwater (Paper 18)
Work Study (Paper 19)

Chairman: Professor J. C. Mitcheson *Rapporteur:* Mr. R. Glossop

681.3:622.7–341.1.094
681.3:622.799.4

Paper 16

Use of a Computer for Grading Iron Ore at Sept-Iles, Quebec

R. W. KIRKLAND, B.E., M.Sc., Ph.D.

R. H. WARING, B.A.Sc., P.Eng.

Terminal Superintendent and Senior Systems Analyst respectively, Iron Ore Company of Canada, Sept-Iles, Quebec, Canada

---

SYNOPSIS

In 1962 the Iron Ore Company of Canada installed an IBM 1620 computer for the grading of direct-shipping iron ores from the Quebec–Labrador iron range. The computer was programmed to cover all aspects of grading, including receipt, blocking and blending of the ore, and final cargo data.

The grading of iron ore, the introduction and use of the computer, and its programs are briefly described.

The computer has proved effective for the grading of iron ore and although there has been no monetary saving it has freed other data-processing equipment from costly interruptions, eliminated tedious copying and hand calculation and offers considerable scope for other applications.

In 1962 an IBM data-processing system was installed and programmed for the grading of direct-shipping iron ore for shipments from the Sept-Iles Terminal of the Iron Ore Company of Canada. Sept-Iles is located on the north shore of the St. Lawrence River 500 miles east of Montreal. The ore is received by railroad from Schefferville, 320 miles to the north in the Quebec–Labrador iron range.

The direct-shipping ore comes from five mines and grade variations, even from the same mine, are sufficient to require blending for uniform, acceptable cargoes. Shipments of bessemer, non-bessemer and manganiferous iron ore are made to Canada, the United States of America, the United Kingdom and Germany with grade variations to meet customer requirements.

The shipping season extends from April to October, the maximum tonnage handled in this period being 13 000 000 t in 1959. Sales of direct-shipping ore for 1963 amounted to 6 015 000 long tons, but were augmented by shipments of concentrates (4 748 000 tons) and pellets (1 923 000 tons) from Labrador City, 200 miles north of Sept-Iles.

The computer was programmed to cover all aspects of grading, including receipt, blocking and blending of ore, and final cargo data. Only statistical control is required for the concentrate and pellets, except when the former is mixed with direct-shipping ore.

MANUAL METHOD OF ORE GRADING

Prior to 1962 the method of ore grading consisted of manually (using desk calculators) blocking the required ore and blending it for dumping to the ship, the necessary conversions and final statistics being completed by data-processing equipment.

As the flow of ore and information are similar with or without the computer, this method is described.

The ore is sampled at the mines in six-car lots of approximately 500 t per sample and is moved in trains of 125 cars to the terminal. The cars are then dropped over a hump and run by gravity over weighscales and into a twelve-track classification yard. The gross weight of each car is automatically punched in a card corresponding to the car by each of the two scales (duplication for accuracy and availability). The allocation of the cars in the classification yard is determined by the grader and is based on chemical analyses.

The cars are removed at the opposite end of the classification yard with the assistance of side-arm pusher locomotives. The sequence of removal from the twelve tracks is predetermined so as to maintain constant the average grade of the cargo. The cars are then pushed into a tandem dumper by a barney hoist, dumped, and the ore conveyed to the ship.

In addition to ore arriving daily from the mines, stockpiles of ore by grade are maintained at Sept-Iles, generally totalling between one and two million tons. Reclaim is by shovel into ore cars, the ore being sampled and the cars handled in a manner similar to those arriving from the mines.

Assuming that sufficient ore is available for a cargo, either in the classification yard, on its way in trains arriving before ship departure, or is being loaded from stockpile, the grader's first step is to choose from the ore available those cars which give the required weight and analyses. For a 30 000-t cargo this involves about 365 cars and eight hours of production. Chemically it is necessary to grade for iron, phosphorus, manganese, silica, alumina and moisture content. Ore type as related to moisture content and screenability also needs to be considered. This information is kept up to date on lists corresponding to each track in the classification yard with the cars shown in the order they stand on the track. The analyses of the cars chosen, plus estimated weights based on previous experience for each mine, are entered on a work sheet.

Using a desk calculator to average the various constituents as the block is built, an acceptable cargo is made.

The next step, blending, consists of making a dumping sequence for the cars chosen so that all the ore loaded into the ship is at the average analyses of the cargo. In practice each 1000 t is average grade.

USE OF COMPUTER FOR ORE GRADING

The initial data-processing system consisted of an IBM 1620 computer including 20 000 units of core storage, console panel and typewriter, a 1622 card-read punch and a 1623 core storage giving an additional 20 000 units. Basically, the system has relatively slow input–output with rapid computation and was designed specifically for technological problems.

Certain ground rules were laid down for programming the computer: (*a*) all aspects covered by other data-processing equipment were to be handled, e.g. conversion of dried analyses to natural analyses, computing net weights from gross and tare weights and final computation of cargo reports; (*b*) programs were to cover the position of cars in the classification yard and the blocking and blending of ore, previously done on a manual basis; and (*c*) print-outs were to be made of all critical data so that it would be possible to return to a manual operation on short notice and also to provide visual control and proper records. Thus a track list is printed for each track in the classification yard, supplying the necessary data for manual grading and forming a record of all ore received. At the end of the blending program a print-out is made of the average analyses of each 1000 t selected to ensure that all requirements are met.

Although nearly two years were spent in investigating the use of computers and the models available, and in programming the system, the initiation was not without difficulties. There is always the problem that the people who understand programming do not understand all the aspects of the work being programmed and vice versa. Simple non-routine items become major when no allowance has been made in the program, examples being the attempt to put a seven-car sample or a plus-125 car train into memory programmed for maximum six and 125. Such items occur once and are corrected, but they cause considerable confusion at the time—especially when operations are tied directly to the grading.

Problems arose because, since the 1620 system was new and with delivery tight and machine-time at other locations limited, the programs were not adequately checked before inauguration. In addition, all the grading staff had to be trained to operate the computer and to cope with stoppages due to incorrect actions. These problems have now been overcome and gradually the staff has become 'computer-orientated'. Three have been trained on programming and, under supervision, can now do additions and modifications to existing programs.

As far as equipment was concerned, two deficiencies soon became apparent: shortage of sufficient core storage and slow output. With only 40 000 units of core storage and a 45 000-t shipment it was impossible to handle any further work until the cargo had been finished and the memory cleared. This was corrected by adding another 20 000 units, bringing the core storage to the maximum. Even with the addition all major programs must be read into the computer at the time they are needed. It has been possible, however, to put some of the shorter programs in memory and to modify others to allow multiple use of the routine without re-entry of the program. This latter capacity is of particular importance for blocking.

Although the computer calculates in milliseconds output was limited, via the typewriter, to 600 characters/min and it was therefore fully occupied when the necessary forms were typed. This was corrected by adding a 1940 serial printer which is five times faster than the typewriter. The typewriter is, however, still required for instructional input.

The use of the computer required a change in communications. Transceivers were installed in the laboratories at the mines and at Sept-Iles and were connected to the grading office. No particular problems were encountered, although it was necessary to train personnel at all locations to operate the equipment and, in some cases, to operate the card punch.

COMPUTER PROGRAMS

As mentioned earlier, there is not sufficient space in the core storage to store the major programs, these being read into memory from cards before each routine, thus setting up the proper sequence. The major routines may be described briefly as follows.

(1) Train data cards showing dried analyses for each sample are fed into the card-read punch, the analyses are converted to natural and new cards with natural analyses are punched. At the same time a train consist giving this information is printed. This is followed by two sets of scale cards which are used for recording the car weights when the train is scaled.

(2) This routine allocates the ore to tracks in memory. Data are read in from the natural analyses cards and the track allocation is done via the typewriter. Finally, track lists are typed corresponding to each track and showing the sequence and grade of ore.

(3) When the scale cards are returned punched with the gross weight of each car, the information is fed into the machine. The weight of each car is compared to the tare weight. The net weight is calculated and assigned to the car in question in memory. Up to this point the machine automatically assigns estimated weights so that blocking can be done prior to train arrival.

(4) Actual blocking of the ore follows with the machine requesting and receiving via the typewriter information as to tonnage and grade required and printing out the selection of ore chosen to meet these requirements. If the result is not satisfactory the grader may change the parameters, erase the block and make another assignment. This happens when the tonnage and grade of the ore in the yard will not make the cargo, in which case ore from stockpile or additional trains from the mines are required.

(5) When an acceptable cargo has been selected the dumping sequence is made so that the ore to ship is blended correctly. A print-out of the analyses of each 1000 t is made.

(6) This program produces the final records covering each shipment.

It will be noted that for most of the programs normal data preparation and statistical programming are sufficient. For blocking and blending, however, it was necessary to use an operations research approach. The programs for blocking and for blending are similar, with the exception that for blocking the required cargo is used for control and for blending the actual blocked cargo is the control. Basically, the computer calculates the accumulated weight × analyses for the six constituents, after selecting each two cars, and compares the results with the required grade × the accumulated weight plus two cars. This gives the ideal grade for the next two cars which would make the actual and ideal blocks identical. The ideal grade is then compared with the actual grade of the next two available cars on each of the twelve tracks and the cars with the minimum deviation (minimum range number) are selected.

$$R_i = \sqrt{\Sigma(A_2 j - A_i j)^2}$$

where R_i is range number for track i; $A_2 j$ is ideal analysis for constituent j; $A_i j$ is actual analysis for constituent j on track i; j is iron, phosphorus, manganese, silica, alumina or moisture; and i is one of twelve tracks.

The range number equation is modified by a scaling factor which compensates for the difference in proportions of the analyses, e.g. iron at 60 per cent and phosphorus at ·060 per cent. The scaling factor for each constituent, K_j, is calculated at the beginning of the program and is then constant. The total weight × deviation of all the unassigned ore in the classification yard from that of the ideal analyses is calculated for each constituent. The scaling factor for each constituent is then the maximum weight × deviation—usually iron—divided by the weight × deviation for the constituent. A tolerance factor, C_j, is used to further modify the range number when a constituent is approaching the allowable limit of deviation from the ideal:

$$C_j = \frac{1}{1 - D/D_m}$$

where D is the difference for a given constituent between the analysis at a given point and the ideal analysis, and D_m is the maximum allowable deviation.

The modified equation for the range number then becomes:

$$R_i = \sqrt{\Sigma C_j \left[K_j \left(A_2 j - A_i j\right)\right]^2}$$

COMMENTS ON COMPUTER OPERATION

After nearly two years of computer-operated grading, together with the improvements in the programs made during this period, it can be demonstrated that the present blocking and blending are superior to work done manually; in addition, tedious copying and hand calculating have been eliminated. Another advantage is that it has been possible to separate grading from other data processing, now done on a larger-type computer (IBM 1401). Grading works on a 24-hour day and 7-day week and hourly interruptions for cargo data from the larger computer would result in an impossible situation.

The introduction of the computer has not brought about any financial saving. There has been a reduction in the labour force, but this saving is offset by machine rental. A definite improvement in the time taken to prepare reports has been obtained and data not previously available are now routine. Blocking is now generally done using actual weights, giving better control on the final cargo.

The use of the computer for assignments other than grading has only recently begun to receive serious consideration. Previously, most of the available programming personnel were used to improve the grading programs. Although further improvements are possible, priority has shifted because of the reduction in sales of direct-shipping ore. Applications made include calculation of ore reserves, current estimated shipping grades and time–material charts for construction. Programs currently in operation include scheduling of mine production and a train-operation simulator which could have application for truck-shovel operations. Highly satisfactory results arising from the future use of the computer on such projects are anticipated.

624.131:622.271 *Paper 17*

Applications of Rock and Soil Mechanics to Surface Mining

A. C. MEIGH, M.Sc.(Eng.), M.I.C.E.
Director, Soil Mechanics, Ltd., London

D. J. HENKEL, Ph.D., B.Sc.(Eng.), A.M.I.C.E.
Professor of Soil Mechanics, Indian Institute of Technology, New Delhi, India

---

SYNOPSIS

The nature and engineering properties of soils and rocks are discussed in general terms, and the factors affecting the stability of excavations are outlined. These include the geological environment, the shear strength characteristics of the overburden and, in the rock, the effect of joints, the shear strength of material in the joints and bedding planes, and the effects of groundwater pressures. Methods of stability analysis are explained.

The importance of adequate investigation of the ground and groundwater conditions is emphasized and the methods used for such investigations are described.

Stability of excavations in rock is discussed in relation to the particular stratification and dip of the strata.

The effects of dewatering, both on stability of excavations and the settlements of compressible strata, are mentioned.

Practical examples are given of the application of soil mechanics techniques to the evaluation of stability of excavation slopes.

The paper concludes that the techniques of soil mechanics can be of great assistance to the planning and economical operation of surface mining.

STABILITY OF SLOPES

The objective in surface mining is extraction of material with the minimum excess excavation. Slopes must therefore be as steep as possible consistent with safety of working.

Similar problems exist in other civil engineering works. Cuttings for roads, railways and canals must, for economy, be as steep as possible without serious risk of collapse over a very long operational life. Short-term stability problems occur when excavations are made for construction of deep basements or other subsurface structures. Here again, slopes must be the steepest possible consistent with safety of working and security of adjacent structures.

In civil engineering the study of the factors governing the stability of slopes forms part of the subject known as 'soil mechanics' while, where rocks are involved, the term 'rock mechanics' is often used. However, the engineering behaviour of all cuttings and slopes in the ground may be embraced in the term 'geotechnics', which is replacing the term soil mechanics.

Stability of slopes is first discussed in terms of slopes in soil overburden. The discussion is then extended to include excavations in rock.

Nature and Engineering Properties of Soils

In civil engineering the word 'soil' is used to describe the relatively weak and unconsolidated materials encountered near the surface of the earth, while the term 'rock' is used for the harder, well cemented materials.

Soils are derived from rocks in a variety of ways, but broadly speaking they can be grouped into residual soils, which are the *in situ* products of the weathering of the underlying rocks, windblown deposits, water-borne sediments, either marine, lacustrine, estuarine, deltaic or fluvial, and glacial soils—boulder clays, moraines, etc.—which were formed by the action of ice during the Pleistocene age. From an engineering point of view there is an important distinction between granular materials (sands and gravels) and cohesive (silts and clays), based primarily on the relative rates at which pore-water pressures are able to dissipate in the respective materials. Another important division is between normally consolidated clays, which have not in the past been subjected to pressures greater than their present overburden pressure, and over-consolidated clays, which have been subjected to higher overburden pressures, and in some cases these pressures were very many times higher than the existing values.

As might be expected from the multitude of ways in which soils are formed, they can be extremely variable in their properties, and an appreciation of their variability and the effects of stratification is essential to an understanding of slope stability problems. In simple cases, however, it is possible to make an analytical approach to the problem, although experience and a detailed knowledge of the local geology also play an important part.

The factors which govern the stability of slopes in soils are the shear strength properties of the soil and the pore-water pressures in the ground. Soil can be considered as a two-phase system consisting of a solid phase, the skeleton of soil particles, and a fluid phase, water plus air in a partially saturated soil and water alone in a saturated soil. It follows that the normal stress across any plane will have two components, one of which is carried by the soil particles and is known as the effective stress, σ', and a fluid pressure or pore-pressure, u, the sum of these constituting the total normal stress, σ. The volume change characteristics and the strength of a soil are controlled by the effective stresses, the pore-pressure being significant only insofar as it determines the magnitude of the effective stress for a given total stress.

The shear strength, S, can be expressed as

$$S = c' + (\sigma - u) \tan \phi'$$

where the cohesion intercept c' and the angle of shearing resistance ϕ' are the strength parameters in terms of effective stress.

Thus, if effective-stress parameters are used, it is necessary to know the pore-water pressures if the shear strength of the ground is to be estimated.

Stability Analyses for Soil Slopes

It is not possible here to give a detailed description of the methods of stability analyses of slopes. In principle, possible failure surfaces are chosen and the conditions of equilibrium of the volume of soil above the failure surface are considered. The ratio of the available shear strength to that required to maintain limiting equilibrium is a measure of the factor of safety of the slope, a factor of safety of unity corresponding to a failure condition. The shape of the failure surface may approximate to a circular arc for a homogeneous soil slope, but will depart from the circular arc to varying extents depending on the stratification and on variations in shear strength and pore-water pressure. In practice, stability analyses are carried out by trial and error processes, using a number of trial surfaces, until the surface giving the lowest value for the factor of safety is found. In recent years the tedium of this process has been reduced by the development of tabular solutions for certain cases[2] and by the use of computer programs.[8]

It must, however, be realized that in many natural strata the stability of a slope may be controlled largely by factors which cannot be handled by analytical procedures. For example, when stratified deposits consisting of fine sands and clays are encountered the seepage of water from the sandy strata may cause internal erosion and the removal of support from beneath the overlying clay strata. These clay strata then fail in tension and the surface layers of the excavation slope may collapse. This failure mechanism is not amenable to analysis, but can be guarded against by the provision of suitable filters to prevent the internal erosion. Many other examples, not amenable to quantitative analysis, occur in nature.

However, when analytical procedures are appropriate, the estimation of the available shear strength in stability analyses may be made either in terms of total or effective stresses. When effective-stress shear parameters are used it is necessary that the pore-water pressures be known at all points along the potential failure surface. The process of excavation will lead to changes in pore-water pressure due to the changes in stress while, at the same time, the equilibrium pore-water pressures will be altered owing to the change in the pattern of groundwater flow consequent on excavation. The time required to establish equilibrium of the pore-water pressures after completion of the excavation will depend on the compressibility and permeability of the soils involved. In coarse granular materials equilibrium will be achieved rapidly, whereas in clays the process may require many months or even years.

The 'long-term stability', that is the stability corresponding to the final pore-water pressures, may be estimated using the effective-stress shear parameters and pore-water pressures corresponding to the final equilibrium groundwater conditions.

In principle, it is possible to estimate the pore-pressures appropriate to the condition encountered in clays immediately after excavation. In practice, however, it is much more convenient to use the fact that in saturated soils no change of shear strength can occur unless drainage and dissipation of pore-water pressure are permitted. This means that the original *in situ* undrained shear strength of the clay C_u may be used to compute the stability immediately after excavation. The method used to examine this condition is called the $\phi = 0$ analysis.

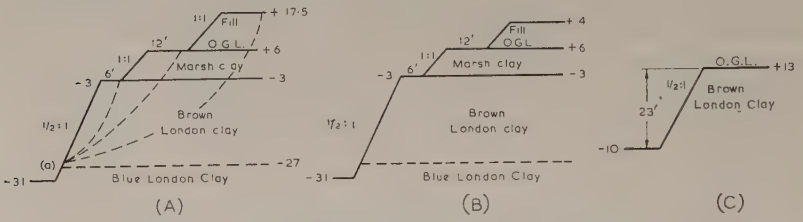


Fig. 1.—Excavation sections in Essex in London Clay.
A—Slip after 1 day; *B*—Slip after 19 days; *C*—Still stable after 4 months.

In general, both the long- and short-term stability should be checked for any excavation but, in practice, the heavily over-consolidated clays are most likely to show a marked decrease in stability with time. The rate of this decrease may be quite rapid in the case of stiff fissured clays, as is illustrated by the following examples. Considerable judgement is necessary if both safe and economical designs are to be produced in these circumstances.

Fig. 1 shows sections through three slopes of excavations in the London clay at a site in Essex.[9] Slope *A*, which has a total height of 48·5 ft, failed within one day of completion of excavation. The first sign of failure was a bulging at the junction of the brown clay and the underlying blue clay, followed by a slip into the lower berm; 4 hours later the slip extended back into the marsh clay and, some 5 hours later, back into the fill. A $\phi = 0$ stability analysis showed that the strength of the brown London Clay necessary for equilibrium was 970 lb/ft^2, compared with an average measured strength of about 1700 lb/ft^2.

The second slope, *B*, which has a total height of 45 ft, also failed, but after a period of 19 days. In this case the strength required for stability was 890 lb/ft^2.

The third slope, *C*, which has a total height of only 23 ft, was still stable after some 4 months; the strength required for equilibrium was only 650 lb/ft^2.

These results show that the reduction in shear strength with time, following excavation in a fissured over-consolidated clay such as the London Clay, may progress rapidly in the early stages.

Investigation Methods for Soil Slopes

Before a stability analysis can be made the geology of the site must be fully understood so that all possible mechanisms of failure can be examined. Shear strength parameters must be assigned to all materials within the possible failure zone, and the groundwater conditions must be established.

Shear strength parameters may be measured in laboratory tests on undisturbed samples obtained from bore-holes or trial pits. Careful methods of sampling are required to ensure that a minimum of disturbance occurs and that the samples obtained are representative of the field conditions. In some materials disturbance may be relatively high with normal sampling methods; and more refined ones, in which thin-walled sampling tubes are used with stationary piston samplers,[3] may be required. In cases of very brittle or friable soils it may be preferable to obtain hand-cut block samples from trial pits.

Groundwater conditions may be investigated using bore-holes and field evidence of seepage. In simple cases groundwater levels can be found from measurements of the head of water in a bore-hole, but in complex ground conditions it is necessary to isolate the particular stratum or zone under investigation from the effects of water in other strata. This is done by placing a porous piezometer tip, surrounded with sand, in a bore-hole and isolating this zone by placing specially designed sealing grouts above, or in some cases above and below, the section of the bore-hole where the water pressure is to be measured.

Permeability, when required, is generally determined from *in situ* tests. These are carried out either by filling up a bore-hole with water and observing the rate at which the water falls (a falling-head test) or by reducing the level of the water in the bore-hole and observing the rate at which the water recovers (rising-head test). In both methods the whole length of the bore-hole can be tested or, by using a system of packers, short lengths of the bore-hole can be isolated in order to test particular strata or zones. This type of test can be carried out in all granular soils, including silts, and in fissured rocks. In clays the times involved are such that testing by this method is not practicable.

Triaxial Compression Tests

The apparatus generally used for measuring the shear strength of soils is the triaxial compression apparatus. For this test a cylindrical specimen of soil is enclosed in a water-tight rubber membrane. The specimen is enclosed in the triaxial cell, in which a fluid can be placed under pressure. An axial load is applied to the specimen, usually at a constant rate of strain, by means of a ram passing vertically through a gland at the top of the cell. In normal testing procedures the stress applied by the plunger is the major principal stress, σ_1, and the cell pressure is the minor principal stress σ_3 (the intermediate principal stress $\sigma_2 - \sigma_3$). The value of the principal stress difference at failure $(\sigma_1 - \sigma_3)_f$ is the diameter of the Mohr circle at failure (Fig. 2). Density and moisture content are also measured as part of the test procedure, and the stress–strain modulus, *E*, is obtained from the initial tangent modulus to the stress–strain curve.

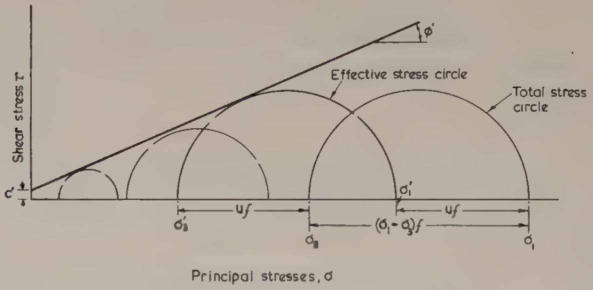


Fig. 2.—Mohr circle diagram.

Connections to the top and bottom of the sample can be made. One of these may be used to control the drainage of the specimen and the other may be connected to a pore-pressure measuring device.

A wide variety of tests may be carried out in the triaxial apparatus, but for normal investigations for slope stability either drained or undrained tests are performed.

In drained tests drainage from the specimen is allowed throughout the test, and the rate of testing is kept sufficiently slow to ensure that full dissipation of any excess pore-water pressure occurs and the effective stresses are measured directly. Undrained tests, in which no flow of the pore fluid from the sample is permitted, may be carried out on undisturbed soil samples to determine the *in situ* strength of saturated clay soils. Pore-pressures may also be measured so that the results are available in terms of effective stress.

The triaxial cell may also be used for the determination of the coefficient of consolidation which controls the rate of pore-water pressure dissipation. These tests are, however, rather complicated, and details are given elsewhere by Bishop and Henkel.[1]

Stability of Rock Slopes

The methods of analysis described for slopes cut in soil overburden have only limited application to slopes in rock, but the principles are similar.

A rock will generally be intersected by a system of joints and bedding planes. The individual pieces of rock between joints and planes may be extremely strong, but the joints and bedding planes will provide surfaces of weakness. Stability is therefore dependent on the shear strength available along such surfaces, on their orientation in relation to the slope and on the water pressures acting across the surfaces.

The shear strength that can be mobilized along a joint surface depends on the frictional properties of the surfaces and the effective stresses which are transmitted normal to the surface. As has already been indicated, the effective stress is the total stress minus the water pressure.

Joint surfaces are not precise mathematical planes, and displacement along joints may involve local shearing through intact rock. In some cases it is possible to evaluate this contribution to shear strength by suitable laboratory or *in situ* testing, but in other cases this is not possible and a conservative value of shear strength must be adopted, without allowing for the irregularities.

The major difference between the analysis of stability in soil and rock slopes is that in rocks the possible failure surfaces are dictated by the orientation and position of joints. A first step is therefore to establish clearly the system of joints and bedding planes. Analysis then is a matter of statics once shear strength parameters can be assigned to the joint surfaces and the water pressure determined.

Another factor to be considered with excavations in rock is the state of *in situ* stress in the rock. The principal stresses in a rock mass and the ratio between the stresses are influenced by the tectonic history of the rock formation. In some cases the principal stresses, particularly the horizontal or near-horizontal, are much higher than the present overburden pressure. Excavation will lead to a release of these stresses and in hard rocks, such as granites and quartzites, fracturing may occur and affect the stability of steep rock faces.[10,11]

Investigation Methods for Rock Slopes

The need for a preliminary geological appraisal has already been discussed in connection with site investigation for soil slopes. With rock slopes it is of paramount importance, since the analysis must be based on an accurate knowledge of the configuration of joints, bedding planes and faults. Subsequent investigation will be by means of trial pits, adits and drill-holes, again with the primary objective of defining the system of joints and the nature of filling material in the joints. In drill-holes, methods using bore-hole cameras or television cameras can be utilized. It is important to have a system for describing the orientation, spacing and thickness of joints. A valuable reference in this connection is John.[6]

Water pressures in joints can be measured by installing observation pipes in drill-holes. The section of rock under investigation is sealed above and below with packers while a small-diameter tube is carried up to the ground surface. The depth of the water-level in the hole may be determined by the use of an electrical measuring device.

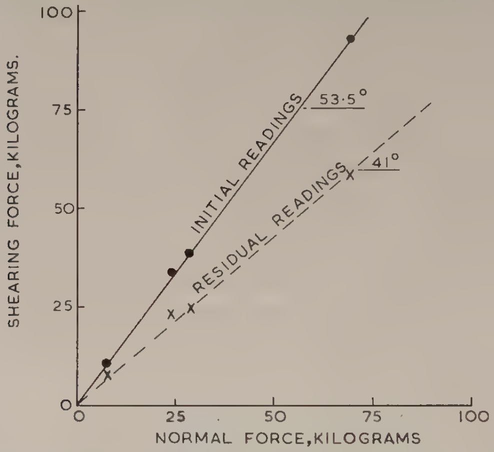


Fig. 3.—Typical result of shear box tests on joint planes.

If large-diameter cores can be obtained with joint surfaces intact, shear strength tests can be carried out in the laboratory. In some cases, however, where the shear strength will depend on the character of the joint infilling, the infilling may be disturbed or washed out by the drilling water, or a soft layer in the rock (a fireclay layer in coal measure shales, for example) may be lost or disturbed in drilling. It is then necessary to obtain block samples for laboratory testing or to carry out shear strength tests *in situ*.

A laboratory investigation to determine the shear strength that could be mobilized along joints in the foundation rock of the Monar Dam has been described elsewhere by Henkel and co-workers.[5] Samples from bore-hole cores which included joints were cut in such a manner that the resulting specimens could be mounted in a shear box with the joint plane horizontal. The entire surface of the joint was immersed in water and the horizontal shearing force required to cause movement for a range of vertical or normal forces was measured. After the initial shear movement had occurred, further displacement took place at a lower value of shearing force, and from the latter part of this test the residual angle of friction could be obtained. Typical results of a series of tests on a specimen are shown in Fig. 3, in which the residual as well as the maximum angle of friction is indicated.

Improving Stability

The stability of slopes in soil and in rock can generally be improved if the pore-water pressures can be reduced. The importance of pore-water pressures in governing shear strength has already been referred to in the section on slope stability. Similarly, the majority of remedial measures for landslides and slope failure consist of, or include, the improvement in drainage. In any particular slope problem consideration should be given to the role of water pressures, either in the soil or rock mass or in fissures and joint planes. Drainage may be of various types: deep drainage, to reduce water pressure within the rock or soil mass, or shallow drainage to intercept surface water. Calculation can be carried out to determine the extent to which slopes can be steepened when drainage is provided and a comparison of costs can be made. The quantity of water which has to be pumped or otherwise removed may be estimated from the results of the *in situ* permeability tests.

In some cases rock bolting can be used to increase the stability of rock slopes. The techniques are similar to those used in roof bolting in deep mining, although grouting around the bolts is more often used in order to increase their working life. It is essential to have bolts to sufficient length to anchor the material beyond any probable sliding surface.

CASE STUDIES

Examples of stability problems in slopes of both soil and rock are given in this section. They are not all concerned with surface mining, but serve to illustrate the principles of slope stability.

Steep Rock Iron Mines, Ltd., Ontario, Canada

The studies carried out in connection with the excavations for the iron mine in Steep Rock Lake are an example of the application of soil mechanics techniques to mining development.[7] The excavations provide useful information on the very flat slopes that are required in soft clay overburden.

The bed of the lake, drained by pumping, consisted of varved clays extending to depths of up to 150 ft. The excavation profile consisted of a series of slopes and berms. The slopes were trimmed to 1 in 3 and the height of each section was limited to 20 ft. The widths of the intervening berms were calculated using shear strengths determined from tests on undisturbed soil samples obtained from borings.

A Landslide in Bristol, Great Britain

A shallow landslide, which occurred following excavations at the toe of a slope in Bristol, illustrates the dominant effect exercised by the inclination of rock strata.

The slope, with an average angle of 31°, rises from Hotwell Road in the south to Southernhay Avenue in the north. At some time in the past the slope was terraced using masonry retaining walls, but the average slope corresponds closely with the dip of the strata of bedrock. In the area the bedrock consists of the Quartzitic Sandstone Group and is made up of a series of layers of very hard sandstone and softer clay shales.

The overall geological picture was obtained from observations of surface exposures and, in addition, soundings were made to determine the depth of bedrock beneath the mantle of fill and weathered material which cover the slopes. A bore-hole was, however, put down to find the level of the rock below the southern edge of Hotwell Road.

The general situation can best be appreciated by reference to Fig. 4, which shows a section through the slope. The rock line shown is the

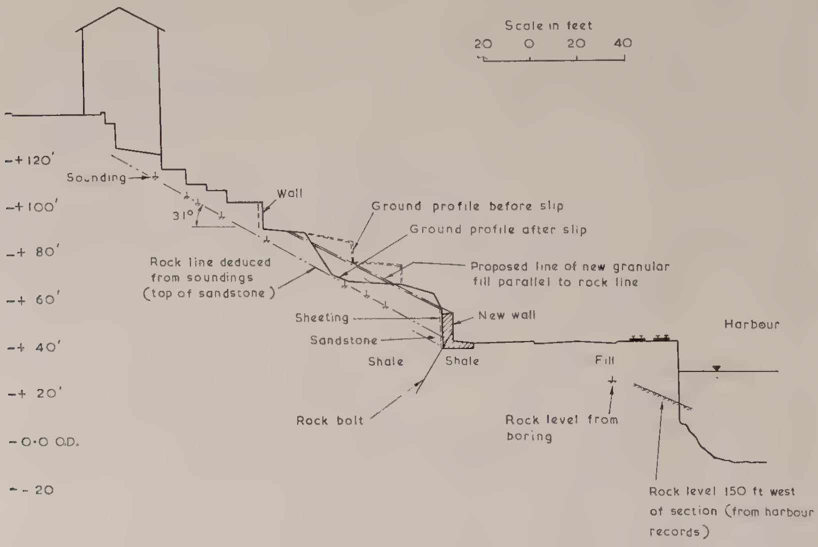

Scale in feet
20 0 20 40
+120'
+100'
+80'
+60'
+40'
+20'
0·0 O.D.
-20
Sounding
31°
Wall
Ground profile before slip
Ground profile after slip
Rock line deduced from soundings (top of sandstone)
Proposed line of new granular fill parallel to rock line
Sheeting
Sandstone
New wall
Harbour
Shale
Shale
Fill
Rock bolt
Rock level from boring
Rock level 150 ft west of section (from harbour records)

Fig. 4.—Section through Bristol landslide.

surface of one of the layers of very hard sandstone which overlies a bed of clay shale. The steep hill slope was probably formed by the process of blocks of an overlying sandstone sliding down on the surface of the weaker clay shale.

When excavations were made at the toe of the slope the weathered material and fill slid down above the sandstone layer. Shear tests were carried out on the weathered material above the sandstone layer and an angle of shearing resistance of 31° was obtained. This value is consistent with the observed movement and shows that the surface layers were very close to a state of limiting equilibrium.

Investigations on the clay shale below the sandstone showed that the angle of shearing resistance was only $21\frac{1}{2}°$ while the liquid and plastic limits were 50 and 25 respectively. If the buttressing effect at the bottom of the slope were removed and the clay shale allowed to soften it was apparent that the layer of sandstone would slide down on the underlying clay shale.

In order to safeguard the slope and the buildings a strong retaining wall was built at the toe. The wall was secured in position by the use of long rock bolts extending below the clay shale into further layers of sandstone. The possibility that the thin layer of sandstone might 'jack-knife' was guarded against by placing granular fill on the ground surface and trimming the slope to an angle of 31°. Great care was taken to drain the fill so that a build-up of pore-water pressure, with the consequent reduction in strength, could not occur.

The level of the rock surface found in the boring near the river showed that the sandstone layer on which the slide had occurred was adequately buttressed against the possibility of sliding into the river.

Slide Movements in the Avon Gorge, Bristol, Great Britain

Sliding along joints or along clay layers in a rock mass is most likely to occur when the joint planes or clay layers dip directly towards a river valley or excavation. Slides can, however, occur in strata containing interbedded clay layers when the dip is parallel to the valley or excavation. Movement directly down the line of dip is prevented by the buttressing effect of the rock overlying the clay layer, but when the rocks are intensively jointed oblique movement can occur. Sliding surfaces develop on the surface of the clay layer as well as on the joint planes in the overlying rock. A recent slip of this kind in the Avon Gorge in Bristol has enabled this phenomenon to be investigated and an approximate method of analysing stability to be developed.[4]

At the site of the slide the River Avon has cut a gorge some 200 ft deep in the rocks of the Carboniferous Limestone Series. The strata dip roughly parallel to the face of the gorge at an angle of 33° and consist of thick beds of heavily jointed limestone separated by thin layers

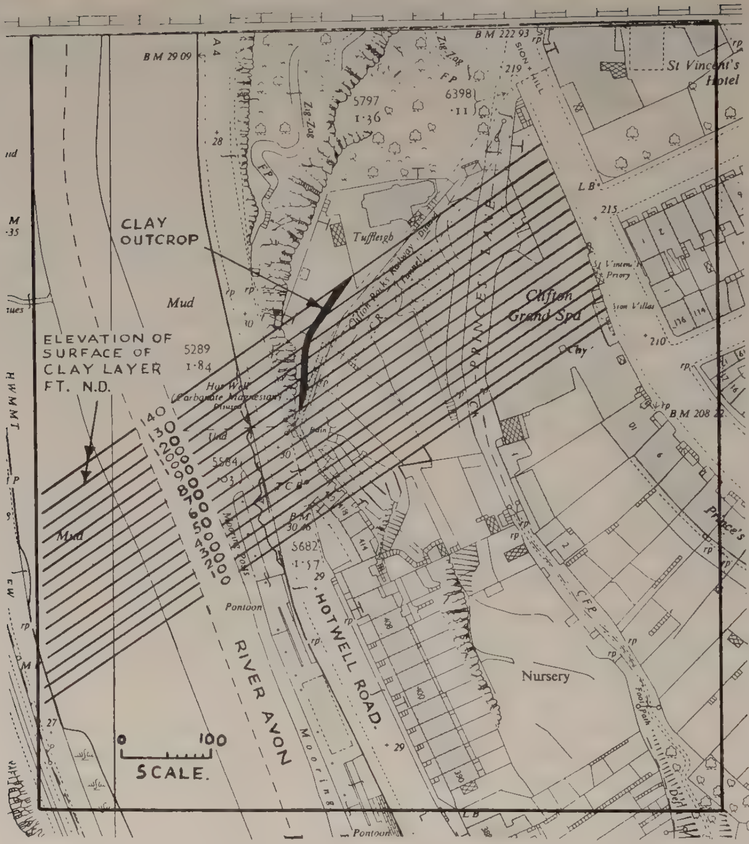


Fig. 5.—Avon Gorge plan.

of very stiff clay. The joint system is normal to the bedding planes and the sides of the joints have been coated with a red clay derived, most probably, from the products of weathering of the limestone together with material carried down from the red Triassic rocks which at one time covered the area.

The lower portion of the gorge has an almost vertical face and the flatter upper parts have been terraced by the construction of a number of masonry retaining walls. A general plan of the area taken from the Ordnance Survey Sheet is shown in Fig. 5.

A feature of the site is the existence of a disused inclined tunnel, which links Hotwell Road on the right bank of the River Avon with the top of Sion Hill. The tunnel portal together with a retaining wall above it extended nearly 50 ft above the level of Hotwell Road and the first evidence of instability in the area was the discovery, at the end of 1956, of cracks between the masonry of the tunnel portal and the face of the limestone cliffs. Further examination disclosed that a crack some four inches wide had opened in the brick lining of the tunnel, cracking and distortion of the steps on the ground surface above the tunnel also being noted. The tunnel portal was shored to prevent further movement, and part of the retaining wall, together with loose fill and rock above the portal, were removed. When this work was completed a layer of clay about 5 ft thick, sandwiched between limestone beds, was found and it appeared that blocks of limestone had been sliding obliquely down the surface of this clay layer.

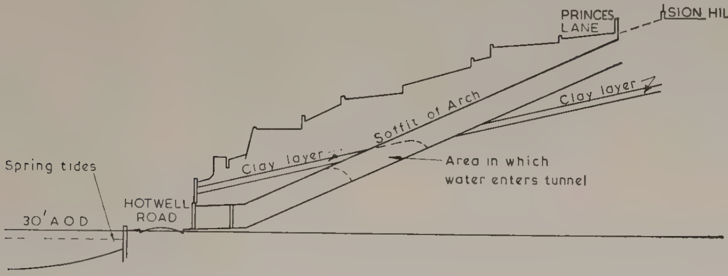


Fig. 6.—Avon Gorge: section along centre-line of tunnel.

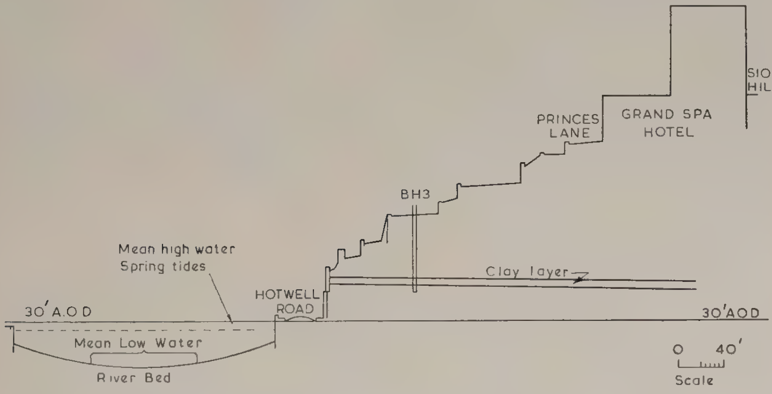


Fig. 7.—Avon Gorge: section at right angles to cliff face.

Figs. 6 and 7 show, respectively, sections along the centre-line of the tunnel and at right angles to the portal and face of the cliff.

Four borings were put down to locate accurately the position of the clay layer, to obtain samples of the clay and to examine the groundwater conditions. These borings showed that the top of the clay layer was essentially a plane surface and its position has been shown in Figs. 6 and 7. The outcrop of the clay layer to the north of the tunnel has been indicated on the plan (Fig. 5) and the series of parallel lines show the elevation of the surface of the clay layer above the Ordnance Datum. The dip of the clay layer was found to be 33°.

As can be seen in Fig. 6, the tunnel intersects the clay bed and at this point a considerable quantity of water entered the tunnel. It was evident that the tunnel was acting as a drain for the water moving through the limestone along the impermeable upper surface of the clay layer. Borings in this area confirmed that there was little or no water above the clay layer.

A bore-hole near the southern wall of the tunnel, about 40 ft from the portal, showed, however, that in the vicinity of the slip the ground water was standing 4 ft above the top of the clay layer. The limestone below the clay was found to be dry and water drained rapidly from the bore-holes once the lower limestone had been penetrated.

The samples of clay taken from the bore-holes showed that the clay was very hard and fissured. The liquid and plastic limits averaged 40 and 22 respectively, while the natural water content was only 15 per cent. Clay samples were also taken from the face of clay exposed above the tunnel portal. The main mass of clay in the exposure had properties similar to those of the bore-hole samples, but it was noticed that the upper $\frac{1}{2}$ in. of clay in contact with the limestone was of a different nature. This thin zone of clay had been weathered and was in fact very similar to the clay which was found in the joints of the limestone. It had a liquid limit of 51 and a plastic limit of 25.

Drained triaxial shear tests carried out on samples from the main mass of clay gave effective-stress shear parameters of $c' = 500$ lb/ft^2 and $\phi' = 22°$. Samples suitable for shear tests could not be obtained from the thin upper zone of material but, by comparison with the results obtained from the main mass of material with a lower plasticity index, it was estimated that the angle of shearing resistance would be about 20°. These angles of shearing resistance are rather low in relation to the plasticity indices of the clays, but they correlate with the results obtained from other clays laid down in Carboniferous times.

For details of the stability calculations, reference should be made to the original paper by Henkel.[4] The calculations show reasonable agreement with the movements observed on the site. It is difficult to estimate the amount of softening or reduction in c' the clay has undergone, but it is considered that the long-term stability when $c' = 0$ can be calculated with some confidence.

Slip at Horsley Opencast Coal Site near Newcastle upon Tyne, Co. Durham, Great Britain

The slip at the Horsley opencast coal site provides another example of instability due to movements along the dip of strata because of the presence of weaker layers. A deep excavation through glacial overburden and the coal measures was being made and a slip occurred.

The site consists essentially of a basin-shaped geological structure and it was in the southern portion, where a deep box-cut was being excavated, that the large slip occurred. Movement started when the excavation was only 35 ft deep and became more severe as the cut was deepened. A section through the slide is shown in Fig. 8.

The general dip of the strata in the area of the slide is about 12½°, while the maximum dip at the top of the slip is about 14½°. These angles correspond to slopes of 1 in 4·5 and 1 in 3·9 respectively.

Careful examination of the coal seams in the excavations showed that, associated with each seam, there was a thin layer of relatively soft clay. The soft layers may well be associated with the concentration of shear strains in these layers during the folding and faulting in the area.

Irrespective of their origin, however, it is clear that these seams of soft material were the major factors in causing the slip. Experience of clays derived from the Carboniferous shales suggests that their angle of shearing resistance will be about 20°. The basin-shaped structure of the site is a classical situation for the development of artesian water pressures and experience on the site confirmed the existence of these water pressures. With the high water pressures and the relatively low angle of shearing resistance, slipping on any dip plane steeper than about 10° might be expected.

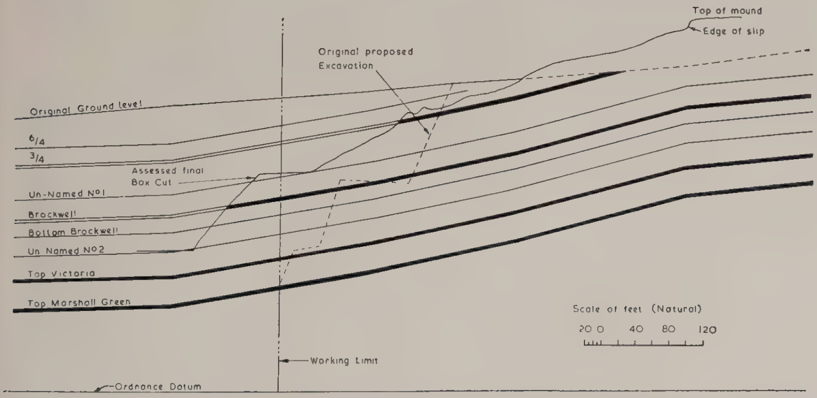


Fig. 8.—Section at Horsley opencast site, showing succession in Coal measures.

TABLE I

Material	Occurrence	Range of natural moisture content per cent	Range of density (lb/ft³)		Atterberg limits (average) per cent		Degree of saturation (average) per cent	Permeability (average) cm/sec
			Bulk	Dry	LL	PL		
Sterile type A	Predominantly in thick bands	15–24	116–124	99–105	49	25	83	4×10^{-6}
Sterile type B	Predominantly in thin bands, interbedded with ore	4–10	106–126	98–116	34	16	25	2×10^{-4}
Specularite ore	Bands, from a fraction of an inch up to several feet thick	3–5	181–192	175–184	—	—	—	—

It is of interest to note that on the west side of the box-cut, where the dip of the strata was at right angles to the cut face, the vertical face over 200 ft high remained perfectly stable.

Masaboin Hill Deep Excavation, Sierra Leone

In 1959 an investigation was made for the Sierra Leone Development Co., Ltd., into some soil mechanics problems in connection with iron ore mining at Masaboin Hill. The principal problem was to select the steepest possible excavation slopes in 'sterile' material to permit economical extraction of the deeper layers of specularite ore. The proposed depth of excavation was in excess of 300 ft.

Masaboin Hill is in the Northern Province of Sierra Leone and is about 60 miles east of Freetown. The hill, which is elongated in shape, about one mile long and half a mile wide, rises above the surrounding coastal plain to a height of about 800 ft above sea level.

The geology of the site is complex. Faulting and local folding are extensive. A general synclinal structure, with the axis running almost north–south, exists over the whole hill.

The hill is generally covered by 30 ft or more of a hard lateritic product with a high iron content referred to as the 'overburden', but which is, in fact, low-grade iron ore, unacceptable as a direct-shipping ore. Surface mining has been in progress since the mine started production about 25 years ago. Over a large area at the top of the hill up to about 120 ft of material has been removed. The bedrock underlying the overburden consists of specularite schist ('powder' ore) alternating with 'sterile' material. The sterile is sometimes concentrated in thick bands. Owing to the uncemented nature of the powder ore, which is generally fine-sand size, and the sterile material, mining is carried out with tractor-scraper units. The sterile material is the product of weathering of a medium-grained and finely bedded micaceous schist, and consists of a compact mass of micaceous silt with a variable content of kaolin.

The sterile material can be divided into two types, depending on its moisture content and plasticity characteristics. The results of various classification tests are given in Table I.

Tests carried out to determine the shear strength characteristics of the sterile material are summarized in Table II.

Some specimens were cut at approximately 45° to the bedding and some parallel. No appreciable difference in strength characteristics could be established. The tests showed that consolidation would take place very rapidly and that the long-term stability condition is essentially the same as the 'end of construction' condition for Type A material. Type B material is stronger under both conditions. For calculation

TABLE II

Sample no.	Type	Undrained triaxial compression test		Consolidated undrained triaxial compression test		Equilibrium shear box test		Drained triaxial compression test	
		C_u lb/ft²	ϕ_u	c' lb/ft²	ϕ'	C' lb/ft²	ϕ'	c' lb/ft²	ϕ'
8	A	1440	18°	1200	22°				
8	A	1400	17°					1150	22°
9	A	1300	15°			1230	20°	1440	21°
12	A	1400	19°	700	29°			1150	20°
13	A					1000	16°	1000	21°
19	A	1400	18°						
16	B	720	26°	800	24°				
17	B							1870	33°
21	B	1700	28°						

purposes the average effective-stress shear strength parameters were taken as:

$$c' = 1150 \text{ lb/ft}^2; \quad \phi' = 21^\circ$$

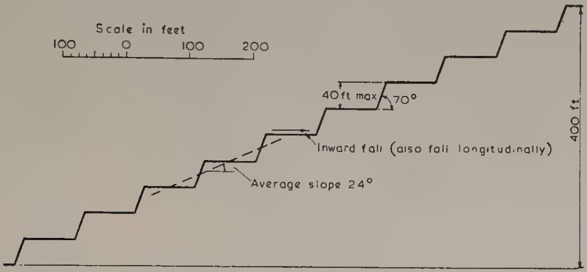
Calculations were made to determine the slope angles which would give a factor of safety of 1·2 for various heights of slope. The results are given in Table III.

TABLE III

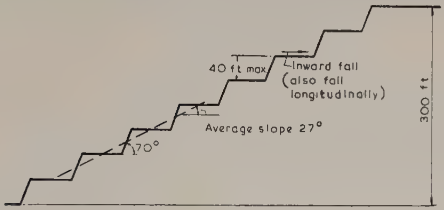
Type of slope	Angle of slope (with horizontal)	Height of slope for $F = 1{\cdot}2$
Steep	90°	40 ft
,,	80°	50 ft
,,	70°	60 ft
,,	60°	75 ft
High	30°	200 ft
,,	27°	300 ft
,,	24°	400 ft

In the calculation it was assumed that there would be no seepage pressures. The actual water-table in the sterile material at depth was not measured, since the investigation was based on information obtained from outcrops and comparatively shallow excavations. However, it seemed extremely unlikely that a water-table would exist within several hundred feet of the surface. Nevertheless, it was proposed to check this in deep drillings which were to be put down at a later date as part of the ore exploration programme.

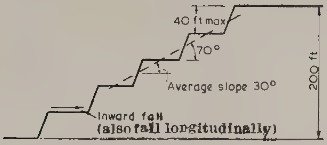
In considering the slope profile to be used it was necessary to consider the erosion problem, the sterile material, particularly Type B, being liable to rapid erosion, and hence giving the possibility of contamination of the ore. A study was made of this aspect, but it is not discussed here.



(a) Overall height 400 ft



(b) Overall height 300 ft



(c) Overall height 200 ft

Fig. 9.—Proposed excavation sections, Masaboin Hill.

In order to minimize erosion it was recommended that benched slopes should be used. Ideally, vertical faces should be used below the benches to reduce as much as possible the area exposed to rainfall and theoretically, as shown in Table III, vertical faces are possible provided that the vertical interval between benches is restricted to 40 ft or less. However, owing to the orientation of some of the bedding planes the use of vertical faces may lead to local slips and it was therefore suggested that, as a compromise, slopes of the order of 70° to the horizontal should be used, with a vertical interval between benches not exceeding 40 ft. It was recommended that the benches should be formed with a longitudinal fall and a cross fall sloping into the hillside. The longitudinal falls will act as drainage channels which will discharge the water to the sides of the hill. As a result of erosion downwash debris will fall on to the terraces. Some of this will be carried away in the drainage system. The excessive accumulation of debris could be handled by the equipment on site. If flow of water on the inside edges of the benches leads to undercutting of the slopes, it will be necessary to provide lined open drains.

The resulting slope profiles for overall heights in the sterile material of 400 ft, 300 ft and 200 ft are shown in Fig. 9.

These recommendations were related to a very long-term mining programme and it will be some years before the predictions can be related to field results. However, it is felt that the information given illustrates the soil mechanics approach to such problems.

CONCLUSION

This paper opened with the statement that the objective in surface mining was the extraction of material with the minimum excess excavation.

Methods have been outlined for the investigation of field conditions and the measurement of the relevant properties of soils and rocks for the assessment of stability. It has been noted earlier that standard analytical procedures are appropriate only in relatively simple geological conditions. Even in these cases, however, great care and judgement have to be exercised to see that the methods of calculation that are adopted cover all the possible mechanisms of failure in the field.

Testing and calculations on their own do not provide a solution to engineering problems, but act only as a guide to judgement and as a means for the organization of engineering experience.

In complex geological conditions extensive field studies and observations both before and during the process of excavation will enable difficulties to be anticipated and sound corrective measures to be devised. In this connection the importance of water pressures in slope stability needs emphasis. High groundwater conditions may seriously reduce the safe slopes in an excavation and in many cases simple drainage works lead to considerable savings.

If drainage by pumping or other means is used to stabilize slopes care must be taken that the adjacent ground is not adversely affected. Consolidation and settlement will occur if the water pressures at the boundaries of compressible soil strata are reduced, and the magnitude of the settlements to be expected should be studied.

The techniques of soil mechanics can be of assistance in the design and construction of economical and stable slopes in surface mining operations. These techniques, however, must be applied with caution, taking into account the overall geological environment.

Acknowledgements.—The authors are grateful to the following for permission to refer to case studies with which they were concerned: City Engineer, Bristol; Lomount Construction, Ltd., and the National Coal Board; Sierra Leone Development Co., Ltd.

REFERENCES

1. Bishop, A. W., and Henkel, D. J. *The measurement of soil properties in the triaxial test.* (London: Edward Arnold, 1957), 192p.

2. Bishop, A. W., and Morgenstern, N. Stability coefficients for earth slopes. *Géotechnique, Lond.*, **10,** no. 4, Dec. 1960, 129–50.

3. Glossop, R., and Meigh, A. C. The influence of soil mechanics techniques on the choice of equipment for sampling and testing. *Final Rep. Public Works and Municipal Services Congr. 1962* (London: The Council, 1962) paper 17, 453–74.

4. Henkel, D. J. Slide movements on an inclined clay layer in the Avon Gorge in Bristol. *Proc. 5th Int. Conf. on Soil Mechanics and Foundation Engng, Paris, 1961.* (Paris: Dunod, 1961), **2,** 619–24.

5. Henkel, D. J., *et al.* Stability of the foundations of Monar Dam. *Proc. 8th Int. Conf. on Large Dams, Edinburgh, 1964.* (Paris: The International Commission . . ., 1964), **1,** 425–41.

6. John, K. W. An approach to rock mechanics. *J. Soil Mechanics Foundation Div., Proc. Amer. Soc. civil Engrs*, **88,** SM4, pt. 1, Aug. 1962, 1–30.

7. Legget, R. F., and Eden, W. J. Soil problems in mining on the Precambrian shield. *Engng J., Montreal*, **43,** Nov. 1960, 81–7.

8. Little, A. L. and Price, V. E. The use of an electronic computer for slope stability analysis. *Géotechnique, Lond.*, **8,** no. 3, Sept. 1958, 113–20.

9. Meigh, A. C. Contribution to discussion on earth dams, slopes and open excavations. *Proc. 4th Int. Conf. on Soil Mechanics and Foundation Engng, Lond., 1957.* (London: Butterworths, 1958), **3,** 255–6.

10. Richey, J. E. Granite, part 4. *Wat. Power*, **15,** Sept. 1963, 374–82.

11. White, W. S. Rock-bursts in the granite quarries at Barre, Vermont. *Circ. U.S. geol. Surv.* 13, 1946, 23p.

551.495:622.271.3
622.848:622.271.3

Groundwater Control in Opencast Mining

S. C. BREALEY, A.R.S.M., B.Sc.(Eng.), M.I.E.E., M.I.Min.E.

Director and Chief Engineer, Powell Duffryn Technical Services Limited, Consulting Engineers; was Project Engineer for the Neyveli Lignite Project

SYNOPSIS

The paper describes the hazards caused by groundwater in opencast mines. Flooding may result from relatively slow seepage or by sudden inundation due to 'piping'. The way in which the stability of the benches and faces is endangered is described with reference to the overburden, the mineral seam, the spoil heaps and the floor of the mine. Soil mechanics considerations are then referred to in respect of the behaviour of consolidated ground and unconsolidated sands, clays, etc., under the influence of hydrostatic pressure.

Groundwater is either free or confined and the properties of aquifers and aquicludes are described. Methods adopted to deal with the problem, such as tunnels, drains and wells, are examined with particular reference to well theory. There are descriptions of field examination by pumping tests, the design of wells, and the instrumentation of the tests. Finally, the paper deals with the interpretation of test results and the design of a system for the mine including the equipment which is required.

Water constitutes a hazard to a greater or lesser degree in all mining operations, and mining below the water-table can only succeed if the water is dealt with effectively. In addition to the more obvious risks associated with flooding, groundwater frequently causes stability problems since the ground near the surface is usually weak. Economic considerations demand the minimum of work in removing overburden and hence the sides of the excavation have to be as steep as possible consistent with safety. In weak ground these slopes are largely determined by groundwater considerations.

Surface runoff must be prevented from entering the excavation, the spoil banks, and the overburden as far as possible, otherwise it will cause trouble directly or add to the groundwater problem. Bore-hole pumps, open drains or tunnels may be used to deal with groundwater and the most difficult situations occur when it is under appreciable pressure. The paper refers in some detail to the pumping tests carried out at Neyveli, South India, in 1956[1] and the solution obtained in the case of this lignite mine where there is a high-pressure aquifer underneath the seam.

STABILITY OF SLOPES

Where the overburden and ore are consolidated rocks there is no stability problem due to groundwater. The faces may be maintained at steep angles and even the rock dumps are not endangered. Although groundwater may be present in quantity, particularly if the rock is fissured or porous, it may be dealt with purely as a drainage and pumping problem without concern for stability failures.

In the case of unconsolidated or weakly consolidated formations, however, such as sands, gravels and clays, stability is the prime consideration. The sheer strength of unconsolidated ground is largely dependent on the effective pressure between the particles, and this pressure is reduced by the hydrostatic pressure, thereby increasing the tendency for slopes to slide.[2]

In the case of spoil banks this situation may be accentuated by the fact that the material is deposited generally in layers parallel to the slope at the angle of repose, thus leaving incipient planes of weakness roughly along the lines of probable slip.

The mixing of beds in the spoil bank may result in better or worse ability to withstand failure than the overburden *in situ*. The spoil would then be less compacted initially and would therefore tend to hold more water. It may also tend to drain less readily if an original permeable bed has been disturbed and mixed with impermeable material, e.g. a bed of gravel overlain by clays. It can be shown[2] in the case of horizontal stratified overburden that the total vertical permeability is less than the total horizontal permeability and therefore water tends to drain horizontally out at the face without building up excessive pressure. However, after mixing in the spoil bank the horizontal permeability will be decreased and the vertical permeability increased. This increases the tendency for water pressure to build up behind the face and also for the water to find its way to the bottom of the heap before flowing out at the toe. A potentially dangerous situation exists if the water-table intercepts the slope line. Bore-holes should be maintained in the spoil heaps so that the position of the water-table is known.

Unless special care is taken, rainwater may gain access to the spoil bank through the surface and through shrinkage cracks, and if drainage is bad the water-table will rise inside the spoil bank. This water, together with the original water, any inflow from adjacent strata and any runoff from the surrounding country, tends to migrate towards the face and if it cannot escape freely it will cause dangerous instability. Enormous slides due to this cause have occurred in the German brown coal mines—in some cases triggered off by vibration from passing trains.

The stability of the overburden faces and spoil heaps may be increased by flattening the slopes or by increasing the number and width of the benches so that the overall slope is reduced, but this has a severe adverse effect on the economics of the operation. Adequate measures to deal with groundwater will, therefore, improve the economics of the operation as well as increasing safety. Neglecting access roads and benches, the volume of a box-cut to expose a rectangular area of seam is given by the following equation:

$$V = A + B \cot \theta + C \cot^2 \theta \quad . \quad . \quad . \quad (1)$$

where V is the volume

A, B and C are constants depending on the width and length of exposed seam and the thickness of overburden

θ is the overall angle of slope to the horizontal of the sides and ends of the excavation.

The shape of the curve V, plotted against θ, is given in Fig. 1 and the effects on V of reducing θ can readily be seen. The effect of the thickness of the overburden is outside control, except insofar as the site of the box-cut can be selected; but the thicker the overburden, the worse the situation.

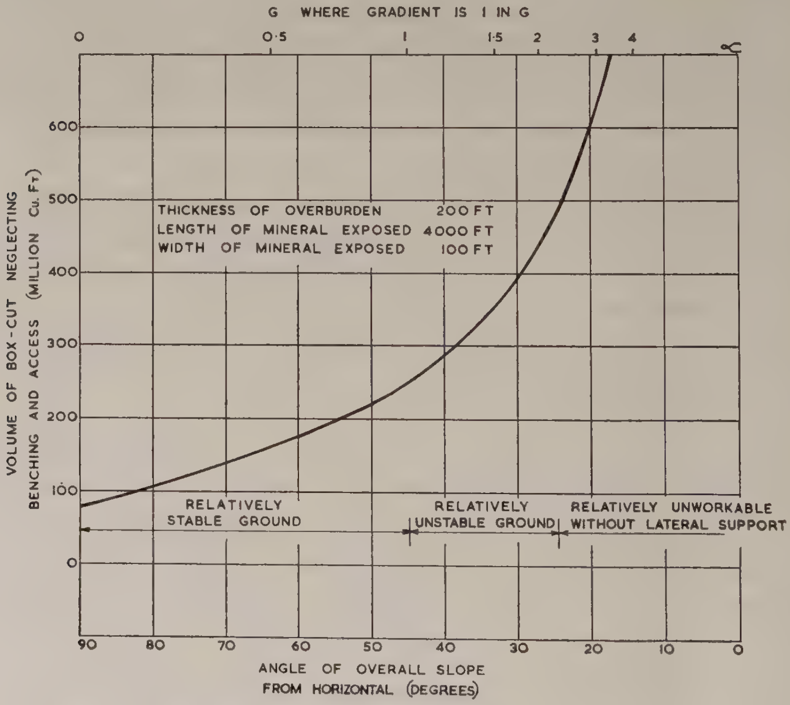


Fig. 1.—Relationship between volume of box-cut and overall slopes.

The stability of spoil heaps can be increased by lowering the groundwater table by drainage, but if the material is not free-draining, e.g. a heavy clay which may also have been puddled by machines, the only practical solution is to flatten the overall slope.

GROUNDWATER

Groundwater is either free or confined in aquifers and its presence may be represented by a water-table or by a pressure surface. Confined groundwater is by definition under pressure and may be artesian, i.e. able to flow at the surface if the aquifer is penetrated by a bore-hole, or sub-artesian. The pressure surface ('piestic' or 'piezometric' surface) indicates the hydraulic gradient and hence the direction of flow in the aquifer.

It is not unusual, in stratified deposits, for a number of free water-tables and aquifers to be present, the former being above the latter (i.e. 'perched'), and this may be misleading if exploration in depth has been insufficient. These bodies of water communicate to a greater or lesser degree depending upon the relative positions and permeabilities of the strata, which are often lenticular in form, and of the recharge and discharge areas.

GROUNDWATER IN THE OVERBURDEN

Free or confined groundwater in the overburden may be dealt with either by allowing it to flow into the mine and then channelling it to sumps or by disposing of it before it flows into the mine. The former method is feasible where quantities are reasonable and where stability and working conditions are not seriously affected. However, in cases where deep excavations have to be made in weak heavily water-bearing ground other measures have to be adopted, the most effective of which is by pumping from bore-holes.

Surface water drainage

Drainage slopes must be maintained on all benches and sumps established at low points to deal with rainfall. Pontoon-mounted pumps are desirable in the main sump if heavy rainstorms are expected. Interceptor drains on the surface outside and uphill of the mine should be so sited as to cut off surface water and channel it to safe outfalls.[3] In the case of long faces it may be necessary to use a siphon to carry water across the mine to an outfall on the worked-out side.

Spoil bank drainage

A system of drains should be established in the floor of the mine which will stay open when covered by spoil, thereby draining groundwater out of the spoil bank when subsequently covered over. In cases where the direction of working is to the rise, a main drain under the spoil heaps may be required to channel this water to a permanent pumping station. Full consideration should be given to the contours at the base of the seam so that an adequate drainage system for the life of the mine can be planned before the box-cut site and the direction of advance are determined. Spoil heap drainage may also be effected by bore-hole pumps if conditions are suitable. By these means the water-table in the spoil may be controlled so as to avoid excessive pressures and thereby prevent slides.

Pre-drainage of the overburden

Where a number of mines are exploiting one field, it is obviously better to plan all pre-drainage operations together in order to achieve mutual assistance and economy. Fig. 2 is a generalized cross-section of part of the Rhineland brown coal basin. The undisturbed free groundwater table is quite close to the surface but, in order to make it possible to work the lignite under 200 m of overburden comprising unconsolidated and highly permeable sands and gravels and clay beds, an extensive system of pumpwells has been developed. These pumpwells are situated in the valley of the River Erft so as to dewater the overburden in the area designated 'new developments' and the resulting cone of depression is indicated.

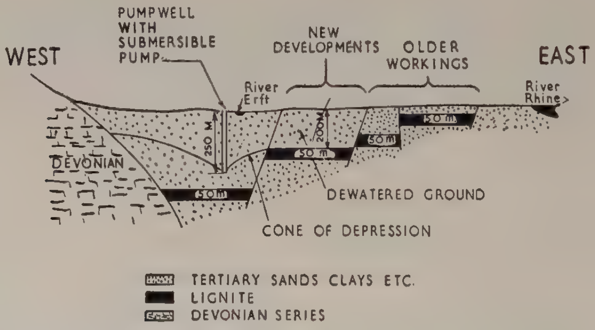


Fig. 2.—Generalized cross-section of Rhineland brown coal basin.

In 1960 over 400 pumpwells were in operation with a total installed capacity of about 50 000 kW pumping about 900 000 000 m^3 per annum, i.e. 28 m^3/s (approximately 1000 ft^3/s). About 12 tons of water were extracted per ton of lignite produced. The water is discharged into the River Erft, or into canals, from where it discharges ultimately into the River Rhine or is tapped off for water supplies. In dewatering operations on this scale the aggregate pumping rate must exceed the rate required to dewater the ground in the time available plus the inflow. When dewatering to the required level has been completed, the pumping rate may be reduced to the inflow rate. Failure of the pumps for a short period will not be disastrous as the voids take time to fill up.

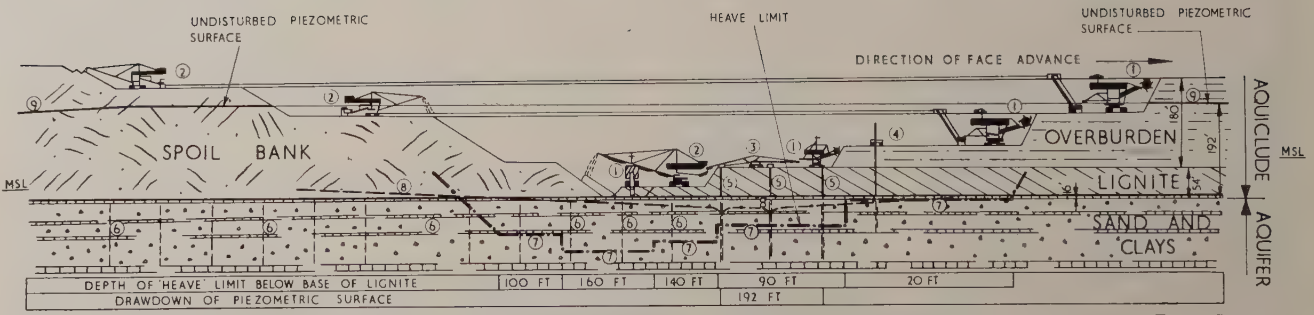
In many of the older mines pre-drainage has been successfully accomplished by driving tunnels horizontally into the lignite seam to intersect bore-holes from the surface. These bore-holes are fitted with screens and gravel-pack filters in the permeable beds to prevent formation material being washed away and to allow the water to drain out. The tunnels are easy to drive in the soft lignite and are supported by timbering. However, the ease with which pumpwells can now be constructed following the advent of reverse-circulation drilling and the submersible bore-hole pump has caused this method to be superseded.

CONFINED GROUNDWATER BELOW THE FLOOR OF THE MINE

Fig. 3 illustrates the condition in the Neyveli opencast mine with a confined-water aquifer below the lignite seam. The aquifer is subartesian and the undisturbed piezometric surface is determined by the water-levels in bore-holes which penetrate to the aquifer (see Fig. 4). The hydrostatic pressure (which acts in all directions) at the top of the aquifer is equal to the height of the piezometric surface above that point (192 ft as shown, i.e. 12 000 lb/ft^2). This is more than balanced vertically before mining by the weight of the superincumbent strata, i.e. 180 ft of overburden at 130 lb/ft^3 plus 54 ft of lignite at 75 lb/ft^3, plus 16 ft of underlying beds at 130 lb/ft^3, which amounts to a total pressure of approximately 30 000 lb/ft^2. If, however, the overburden and lignite are removed by mining operations, the balancing pressure is reduced to that of the 16 ft of the underlying beds only, i.e. approximately 2000 lb/ft^2, leaving an out-of-balance upthrust of 10 000 lb/ft^2 under the floor of the mine.

Heave

Unless the floor is very strong, as a beam, it would 'heave' upwards under this pressure and eventually fail, thus providing a path for the inflow of groundwater into the mine. If the transmissibility and the hydrostatic head are high, this flow will be extremely rapid and cause serious flooding within minutes. The heave limit-line, shown in Fig. 3, indicates the level where the hydrostatic uplift is just balanced by the weight of ground above. It is not necessary to deal with aquifers below this line provided that the water pressure in them is no greater than that in the aquifer shown—a most unlikely situation. In order to protect the mine completely against heave it is necessary to create a drawdown sufficient to restore the balance of forces.


UNDISTURBED PIEZOMETRIC SURFACE
HEAVE LIMIT
UNDISTURBED PIEZOMETRIC SURFACE
DIRECTION OF FACE ADVANCE
①
②
②
③
④
⑤
⑥
⑦
⑧
⑨
SPOIL BANK
MSL
OVERBURDEN
LIGNITE
SAND AND CLAYS
180'
192'
54'
AQUICLUDE
AQUIFER
MSL
DEPTH OF 'HEAVE' LIMIT BELOW BASE OF LIGNITE
100 FT
160 FT
140 FT
90 FT
20 FT
DRAWDOWN OF PIEZOMETRIC SURFACE
192 FT

STRATA	DESCRIPTION	AV DENSITY	LBS/CU FT	TONS/SQ FT/FT DEPTH
	SPOIL AND STRATA	2·09	130	0·0583
	LIGNITE	1·16	75	0·0324
	WATER	1·00	62	0·0279

THICKNESS OF OVERBURDEN — — — — — — — — — 180 FT
THICKNESS OF LIGNITE — — — — — — — — — — 54 FT
DEPTH OF UNDISTURBED PIEZOMETRIC SURFACE BELOW GROUND LEVEL — — — — 60 FT
HEIGHT OF GROUND SURFACE ABOVE MEAN SEA LEVEL — — — — — — 160 FT

1. Bucket-wheel excavator
2. Spreaders
3. Mobile transfer conveyor
4. Well-drilling rig
5. Operating pumpwells
6. Sealed and abandoned wells
7. Danger of 'heave' limit —·—·—
8. Drawdown due to pumping - - - - - -
9. Undisturbed piezometric surface ————

Fig. 3.—Cross-section of Neyveli lignite mine showing limit of danger of heave and hydrological conditions.

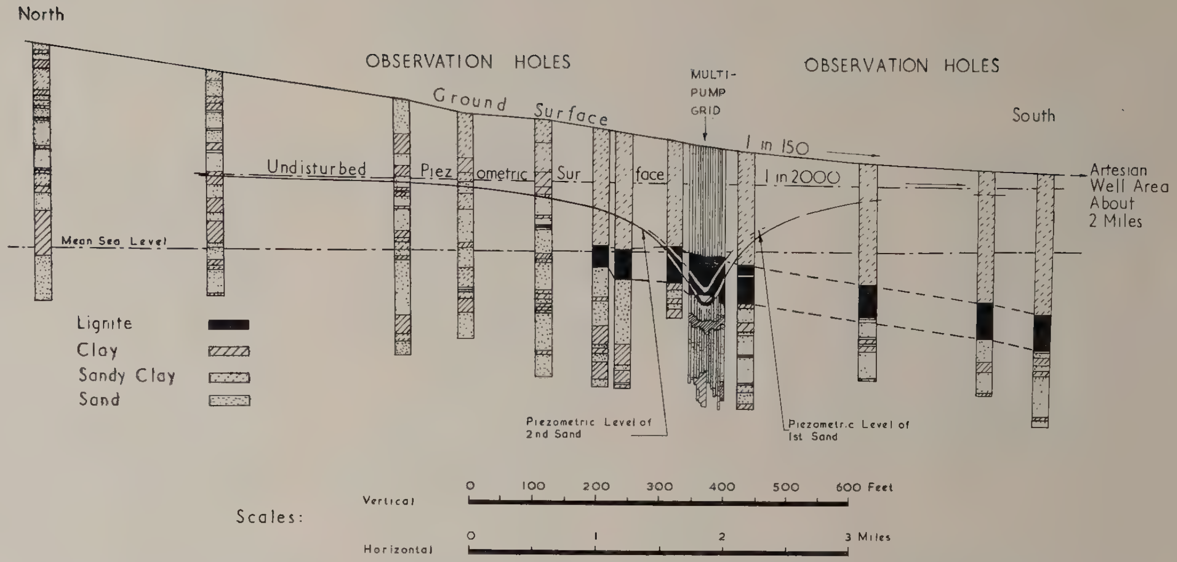


Fig. 4.—Section from north to south of the Neyveli lignite field showing drawdown during multi-pump test.

Piping

Another hazard is 'piping', a phenomenon occurring where the confining beds do not provide a perfect seal. For instance, local thinning out or transition into more permeable material is common. Thus an upward path, however difficult, is provided for the groundwater. The resulting flow may be small initially, but it rapidly increases as first the smaller and then the larger particles are washed out. Old bore-holes can provide the necessary initial path if not effectively sealed off (not an easy matter when the head is high). These phenomena not only result in disastrous flooding, but may also cause collapse of the benches or of the spoil bank, thereby damaging the machines. In order to protect the mine completely against piping, the groundwater pressure surface must be lowered to a level below that of the mine floor.

GROUNDWATER FORMULAE

Hydrologists have developed a number of formulae to serve as guides for the field testing of aquifers and the interpretation of the results. Darcy's Law[4] is the basis and it states that the flow rate through a porous medium is proportional to the head loss and inversely proportional to the length of the flow path. From this law idealized formulae have been developed relating groundwater flow to the height of the water-table or pressure-table. In the case of unconfined groundwater an exact formula cannot be obtained even for an idealized model, but approximations have been suggested by Dupuit, Forchheimer and others.[4]

In the case of confined groundwater the Thiem formula[5] presupposes an equilibrium condition and the Theis formula[6] a non-equilibrium condition, i.e. time is a factor. The Theis formula and its variations are the basis of modern hydrological work; it is stated as follows:

$$s = h_0 - h = \frac{Q}{4\pi T}\int_u^\infty \frac{e^{-u}\,du}{u} \quad . \quad . \quad (2)$$

where s is the drawdown at the point under consideration

h_0 is the height of the piezometric surface at the point, before pumping begins

h is the height of the piezometric surface at the point, after time, t

Q is the total rate of radial flow to the well

T is the coefficient of transmissibility of the aquifer

e is the base of the Napierian logarithm

u is the expression $\frac{r^2S}{4Tt}$

where r is the radial distance of the point from the well

S is the storage coefficient

t is the elapsed time after pumping begins.

Using field values, expression (2) reduces to:

$$s = \frac{114{\cdot}6Q}{T} \quad W(u) \quad . \quad . \quad . \quad . \quad (3)$$

where s is the drawdown in ft
Q is the flow in U.S. gal/min
T is the transmissibility in U.S. gal/day/ft

$W(u)$ is known as the 'well function' and is the integral expression in (2). (Published tables of values are available.[7])

$$u = \frac{1{\cdot}87\, r^2 S}{Tt} \quad . \quad . \quad . \quad . \quad (4)$$

r is the radial distance in ft
S is the storage coefficient, dimensionless (usual range 5×10^{-5} to 5×10^{-3})
t is the elapsed time in days.

Using Imperial gallons in Q and T, the expression for s is the same as (3), but for u it becomes:

$$u = \frac{1{\cdot}557\, r^2 S}{Tt} \quad . \quad . \quad . \quad . \quad (5)$$

Methods of solving for S and T may be found in works on hydrology[4] and curves can then be plotted of drawdown against distance at a given time (i.e. the profile of the 'cone of depression') and drawdown against time at a given point.

The Jacob Modification Formula[8] is an approximation and is as follows:

$$s = h_0 - h = \frac{2{\cdot}30\ Q}{4\pi T} \log\left(\frac{2{\cdot}25Tt}{r^2 S}\right) \quad . \quad . \quad (6)$$

(U.S. units)

The expression in brackets equals $0{\cdot}56/u$. Using semi-logarithmic plots this formula gives straight-line relationships between s, Q, t and r and these relationships may be used directly for the solution of mining problems. The formulae are, of course, true only for idealized conditions which are not present in practice, but they are nevertheless of great value in interpreting data obtained from field tests.

The Theis formula can also be used for free groundwater conditions, although it is invalidated near pumping wells where the flow is convergent in the vertical plane instead of purely radial as in the case of a confined aquifer. Partial drainage of aquifers, which may be necessitated by mining operations, also affects the results for the same reason. especially within the drainage zone.

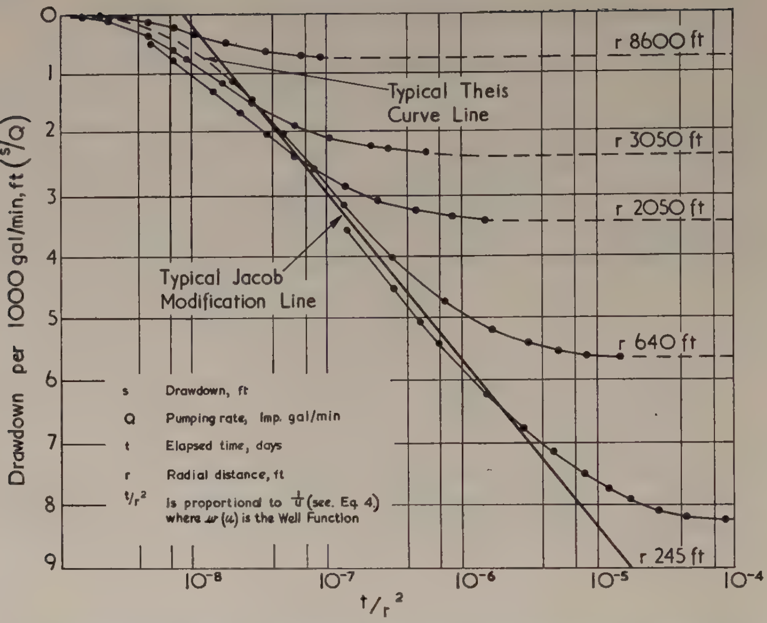


Fig. 5.—Multi-pump test at Neyveli (pumping 22 000 gal/min): semi-log plot showing deviation from Theis theory due to leaky aquifer conditions.

An interesting variation is the ‘leaky’ aquifer model developed by Jacob[9] and in the author’s opinion this is often a better approximation to field conditions, particularly where aquifers and confining beds are known to be lenticular and where the groundwater is not therefore fully confined. The typical shape of the curves is shown in Fig. 5, these indicating the establishment of near-equilibrium conditions, which is contrary to the predictions of the Theis formula.

Boundaries

The Theis theory presupposes that the aquifer is unbounded, but in practice there are, of course, finite boundaries. Provided these are far distant their influence will be small, but nearby boundaries considerably affect the drawdown–distance and drawdown–time curves. One common ‘boundary’ is where the aquifer is intersected by a river, i.e. a source of recharge. In this case, and provided that the river is not so insignificant as to be dewatered, the ‘drawdown’ at the river, due to a pumping well some distance away, will be virtually zero. The drawdown curves can then be constructed by the method of images.[4] In this method the boundary is replaced by an imaginary recharge well of the same capacity as the pumping well at an equal distance on the far side of the boundary. A similar method is used in the case of an impermeable boundary, except that an imaginary pumping well is then used. Fig. 4 indicates one 'boundary' at Neyveli, i.e. the artesian well area to the south.

One difficulty arises from the fact that a boundary may not become immediately apparent as the radius of influence of pumping increases only slowly after the initial period. It may eventually show up as a bend in the drawdown–time curve and this illustrates the advisability of continuing pumping for a reasonably long period of time. Nevertheless, impermeable boundaries will be favourable in the context of this paper as the drawdowns will be greater than predicted, and permeable boundaries will not be unfavourable if the long-term increase in drawdown predicted by the formula is ignored, e.g. the pumping rate as calculated will be more than enough to ensure the fixed drawdown required to protect the mine.

METHODS OF DETERMINING THE GROUNDWATER HAZARD

An essential preliminary of all mining operations is to acquire an adequate knowledge of the physical structure of the rocks encountered, and while this information is being obtained evidence is accumulated concerning the presence of groundwater, e.g. in bore-holes. At this stage, judgement must be exercised as to whether special groundwater investigations are required. Quite frequently, of course, general experience (as distinct from ignorance) may justify adopting the attitude of 'letting the water run into the hole and then pumping it out'.

Circumstances which should lead to a decision to undertake groundwater testing can be summarized as follows:

(*a*) high rainfall and the resulting topography in the neighbourhood;
(*b*) experience in mining or civil engineering works in the same or similar locations;
(*c*) the presence of waterwells in the area, particularly artesian wells;
(*d*) the nature of the formations as regards consolidation, size grading and water content;
(*e*) the behaviour of the formations during drilling, e.g. water-levels, loss or gain of circulation, behaviour of bore-hole walls.

If these indications are unfavourable, failure to undertake groundwater investigations at an early stage will certainly result in delay and expense later, in addition to the difficulty in predicting mining costs. In many mining operations it is as vital to obtain groundwater information as it is cores of the strata.

The strata should be tested well below the base of the mineral deposit for the presence there of confined groundwater, the lower limit being determined by the possibilities of heave and piping.

During these operations the bore-holes themselves should be studied from the point of view of groundwater as their presence later could be an embarrassment; they provide ready communication between aquifers and pierce confining beds and this can have serious consequences. In some countries it is, in fact, illegal to connect aquifers or to allow surface contamination of groundwater. In any event, it is desirable to plug the holes below the orebody with concrete after use.

Exploratory drilling

During drilling operations water-levels must be carefully recorded and some indication obtained of the presence of aquifers with cores if possible or at least formation samples. Geophysical methods can also be used. Down-the-hole pressure-gauges can only be used in open hole where the walls will stand. Percussive drilling in unconsolidated ground has the disadvantage of requiring the use of casing, and mudflush rotary drilling is undesirable as it seals off aquifers, thereby making data unreliable and difficult to obtain.

Reverse-circulation drilling using water only as the flushing medium is recommended; reliable samples are obtained and permeable formations are not sealed off, so making subsequent pumping operations easier; also large-diameter holes can be drilled rapidly with this method without casing in the type of weak ground where groundwater problems are the most serious.

Formation sampling

Samples obtained by reverse-circulation drilling consist of lumps of clay and particles of sand and gravel, the latter usually indicating aquifers. Particle-size distribution curves should be made and from these an indication of the permeability can be obtained and suitable well-screens specified. The '*in situ*' permeability cannot be accurately obtained, although Hazen[2] has developed an empirical formula for the permeability of uniformly graded sands in the loose state as follows:

$$K = CD_{10}^2 \quad . \quad . \quad . \quad . \quad . \quad (7)$$

where K is the permeability, cm/s

C is a constant, 100–150, dimensions 1/cm sec

D_{10} is the effective grain size, cm, i.e. 10 per cent of the grains are smaller than this size.

Pumping tests

The only satisfactory method of determining the characteristics of an aquifer is by pumping tests. The procedure is to construct one or more

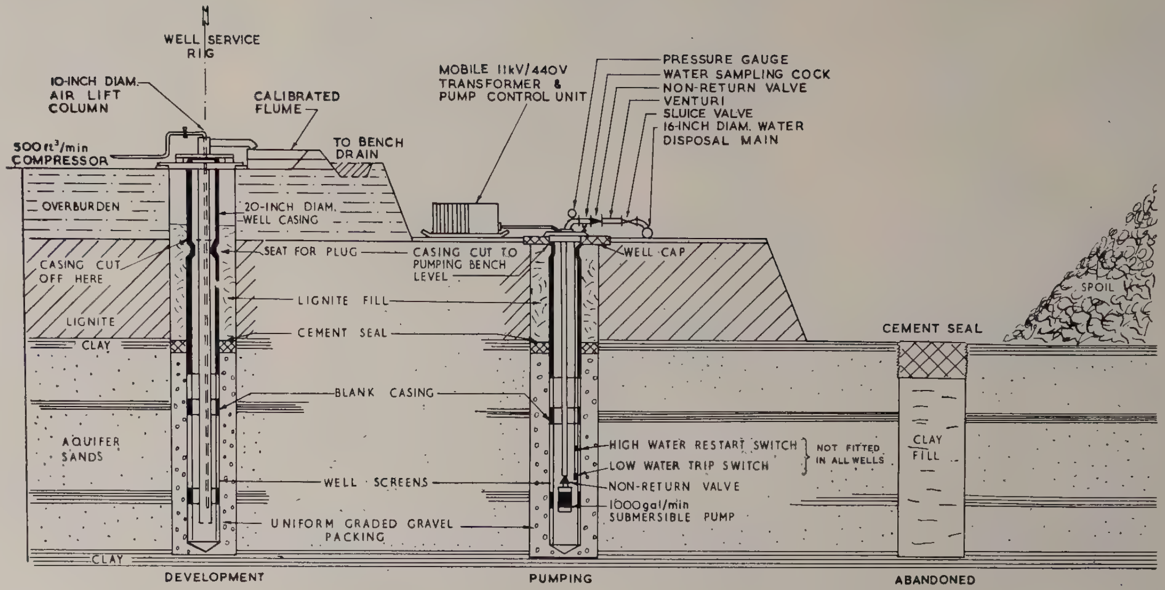


Fig. 6.—Sequence of operations for pumpwells on benches in Neyveli lignite mine.

pumpwells and an array of observation holes in which water-level readings can be taken. Preliminary small-scale tests are usually necessary to obtain approximate values of S and T and by means of formulae 2 and 6 determine a suitable rate of pumping, the well-pattern, the observation-hole pattern, the duration of pumping and the times at which to take water-level observations.

If possible, the drawdown achieved should approximate to that which will eventually be required, and a sufficient time should be taken to achieve a degree of stability and a zone of influence of reasonable size. This is particularly so if departures from the idealized conditions are known or suspected, as in these cases the results of small-scale tests cannot be extrapolated safely.

Pumpwell and observation-hole construction

Unless reasonable care is taken in the construction of pumpwells and observation holes considerable trouble can be experienced with the pumps and faulty data may result. The pumpwell construction must ensure that sand is excluded from the pumps and either the filter-well or the non-filter-well principles may be used. The former employs a wire-wound screen, a graded gravel-pack or a bonded gravel filter pipe, the passages being so sized as to exclude most of the formation particles. The latter principle requires a larger-diameter hole with a simple perforated casing surrounded by a coarse gravel-pack. The velocity of flow into the well is thereby made so low that sand particles are not carried into the pump. Fig. 6 shows this form of construction, the cost being about £7 per ft of completed well. With reverse-circulation drilling these wells are easily and rapidly constructed and are ideally suited to mining work. Observation wells are of smaller diameter, but effective communication with the aquifer is essential. Before use the wells should be cleaned out by over-pumping, for which an air-lift pump is very convenient.

Pumps, etc.

Submersible bore-hole pumps are very convenient for mining purposes as they are cheap and easy to handle, although shaft-drive pumps may be preferred for more permanent locations. Water-levels are measured by drum-type clockwork chart recorders or by hand-plumbing, so that a drawdown–time plot can be constructed for each location. Constant flows must be maintained and the usual methods of flow-measurement may be used, i.e. V-notches, weirs and orifice meters.

Pumping test procedures

Large-scale pumping tests have to be carefully planned if they are to be successfully conducted and yield accurate results, and preliminary tests are necessary for this purpose. Fig. 7 illustrates the pattern of pumpwells and observation holes used at Neyveli. The pumpwells were concentrated as near as possible to the site of the box-cut and lines of observation holes were drilled in the quadrant of the lignite field which was to be worked first.

All the wells must be thoroughly developed and tested and all other preparations made well before the main test is due to begin. Then a period of quiescence must be allowed so that the piestic surface may stabilize to the 'undisturbed' condition, this state being verified by repeated plumbing. This condition at Neyveli is shown in Fig. 7, the distortion being due to a flowing bore-hole in the old quarry which had not been sealed off. The general gradient in the southerly direction is due to the discharge of a large number of agricultural artesian wells a few miles away.

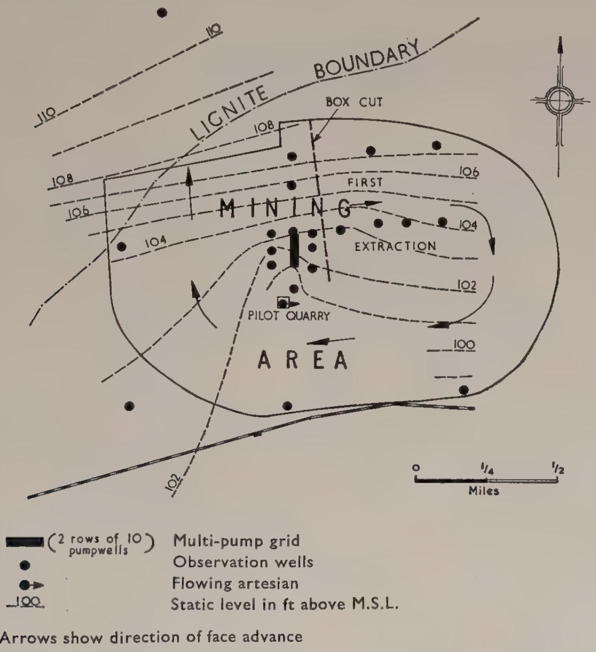


Fig. 7.—Layout of multi-pump test grid and contours of piezometric surface prior to pumping at Neyveli.

At the time selected, an audible 'start' signal is given and all the pumps are started simultaneously and their discharges adjusted to the predetermined quantities. If the electricity supply system is limited in capacity, it may be necessary to stagger the start of the pumps, but this invalidates the early readings and makes awkward the selection of the time-scale zero. Similar procedures should also be followed on shutdown and recovery curves obtained until near-equilibrium conditions are again reached, i.e. the original 'undisturbed' piestic surface is restored. This period may give steadier readings as it is not subject to irregularities in pumping.

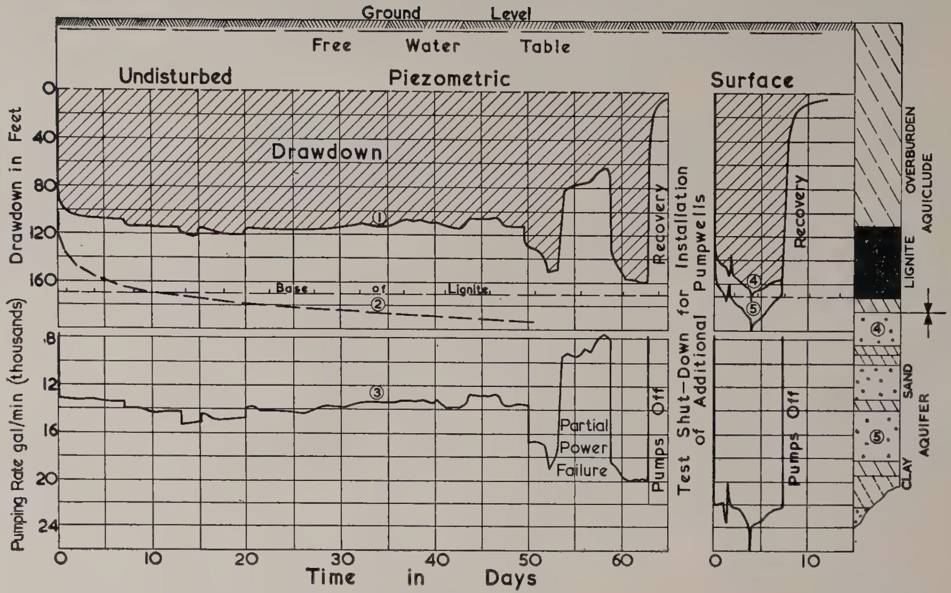
It is useful to carry out a test in stages in order to confirm that the addition of a group of wells to those already pumping results in the drawdown being equal to the sum of the drawdowns when each group is taken separately. This principle is known as the 'principle of superposition'; it is very important in designing the groundwater control system. It is true in time and space, provided that the aquifer does not suffer a permanent 'set' or deformation. (Aquifers are 'elastic' within limits.)

Interpretation of results

The mass of data accumulated during the tests presents a formidable problem in processing into a form convenient for interpretation and for planning the groundwater control system. Calculations may be made to determine S and T in the formulae but, owing to the approximations assumed by the theory, considerable deviations from the theoretical may be expected. Both the Thiem formula and the Jacob modification of the Theis formula indicate a logarithmic relationship between drawdown (s) and distance (r); therefore, if s for a given pumping rate is plotted against log r for each observation hole, a curve approximating to a straight line should be obtained and this may be used for planning the water-control system directly. A sufficient time must, of course, have elapsed for near-equilibrium conditions to be attained, i.e. for t no longer to influence the drawdown. The results obtained from such a test give the relatively short-term situation, but nevertheless one which has to be provided for when mining starts. Long-term effects will invariably be more favourable as drawdown tends to increase with time, even if very slowly, and the pumping rate may be reduced accordingly as mining operations proceed.

THE NEYVELI LIGNITE PROJECT

The Neyveli lignite project presented one of the most difficult groundwater situations ever encountered in opencast mining. Fig. 3 shows the conditions, i.e. a high-pressure sub-artesian aquifer of high transmissibility (approximately 150 000 Imp. gal/day/ft) situated just beneath the lignite bed. It was therefore decided, after preliminary pumping tests, to carry out a large-scale pumping test, and in view of the evident departures from idealized conditions it was considered that the full drawdown should be attained at least over a small area.



1. Drawdown at centre of pumping grid
2. Theoretical drawdown based on results of single pump tests at pump grid (non-equilibrium)
   - *Q* 14 000 gal/min
   - *T* 100 000 gal/day/ft
   - *S* $2{\cdot}4 \times 10^{-4}$
3. Rate of pumping
4. Drawdown recorded at observation well in first sand (centre of grid)
5. Drawdown recorded at observation well in lower sand (centre of grid)

Fig. 8.—Rate of pumping during multi-pump test at Neyveli and resulting drawdown at the centre of the pump grid compared with that predicted from preliminary tests.

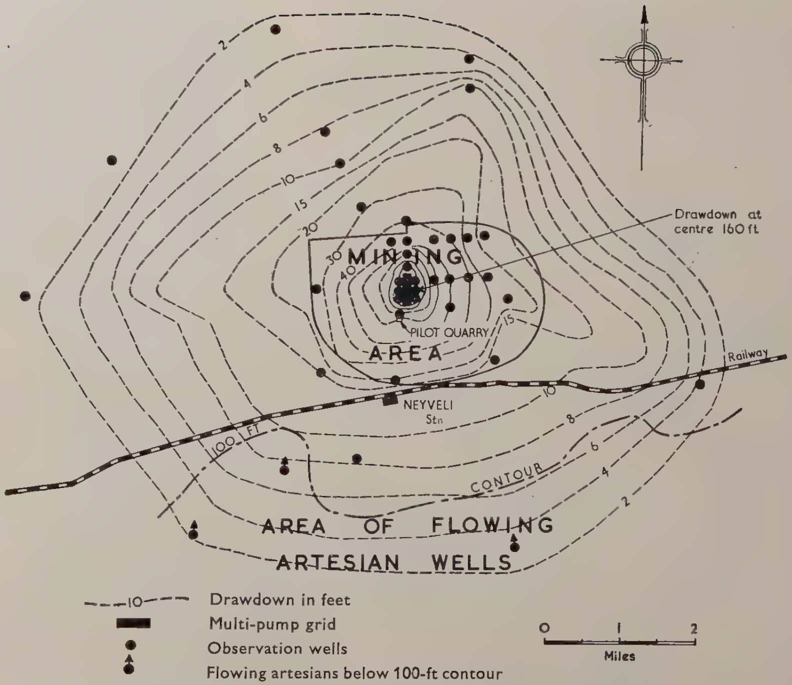


Fig. 9.—Drawdown achieved at Neyveli after 63 days' pumping; near-equilibrium conditions at pumping rate of 20 000 gal/min.

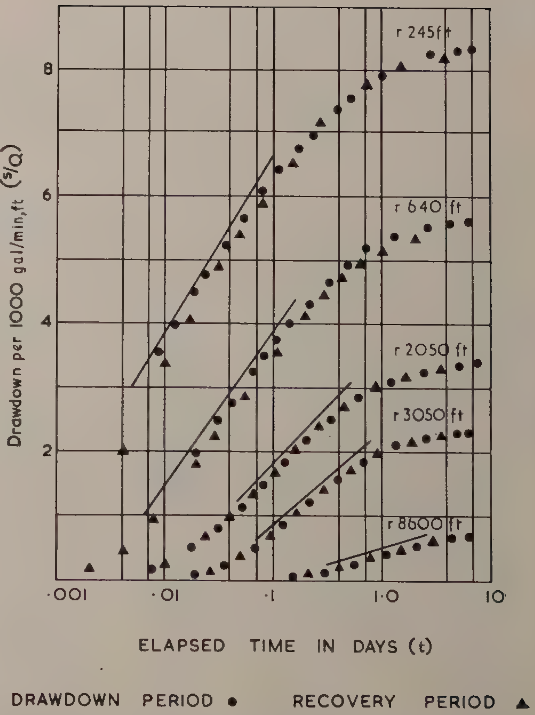


Fig. 10.—Semi-log plot of multi-pump test at Neyveli showing reduction of drawdown rate with distance and 'leaky' deviation from straight lines.

Fig. 8 shows the rates of pumping and the drawdown achieved at a recorder hole situated at the middle point of the pump grid during the course of the main test, i.e. the point of maximum drawdown. The departure of the drawdown curve from that predicted by the Theis formula and the results of the preliminary tests is illustrated; this clearly demonstrates the danger of extrapolating small-scale tests too far. Fig. 4 shows the cross-section of the deposit and the cone of depression in the pressure surface in two sand beds due to continuous pumping for 63 days. Fig. 9 shows the resulting contours of the piezometric surface. The flows from the existing artesian wells shown in this diagram were affected by the drawdown.

Fig. 5 illustrates how the actual curves of s/Q plotted against t/r^2 (log scale) for a number of observation holes are similar in shape to those predicted by Jacob in his leaky aquifer theory and how they differ from the Theis and Jacob modification curve shapes. Fig. 10 shows drawdown–time curves for a number of observation holes. These curves are S-shaped, whereas true Theis curves would be straight lines with a curved portion near the origin. The slopes of these straight lines would be a measure of the transmissibility, T. The curves not only indicate a deviation to a near-equilibrium condition, i.e. s constant, but also a gradual increase in transmissibility (as obtained from the slopes of the straight portions of the curves) as the radial distance increases. From geological considerations alone this is most unlikely to be true, but 'leaky' conditions would give this effect by reducing the radial flow progressively as r increases and hence reducing the hydraulic pressure drop, i.e. drawdown. Also, the time at which near-equilibrium is reached increases with radial distance, i.e. the 'cone of depression' continues to expand after near-equilibrium has been reached at the centre.

Fig. 11 shows the drawdown–distance semi-log plot for near-equilibrium conditions, and the predicted straight-line characteristic is confirmed. The scatter of the points up to 800 ft radial distance can be explained by the fact that these points were near the asymmetrical pumping grid and some were actually inside it and there was, of course, considerable deviation from the symmetrical radial flow pattern required by theory in this region. Also, the pumpwells did not fully penetrate the aquifer and there must have been some vertical convergence of the flow near the wells.

Fig. 11 also proves the principle of superposition as the points for different total pumping rates fall on the same curve, i.e. the drawdown per 1000 gal/min pumped was the same, irrespective of the pumping rate. Fig. 11 was used to plan the groundwater control system for the mine directly, without resorting to any of the formulae. It can be used to predict the near-equilibrium drawdown at any point, due to the

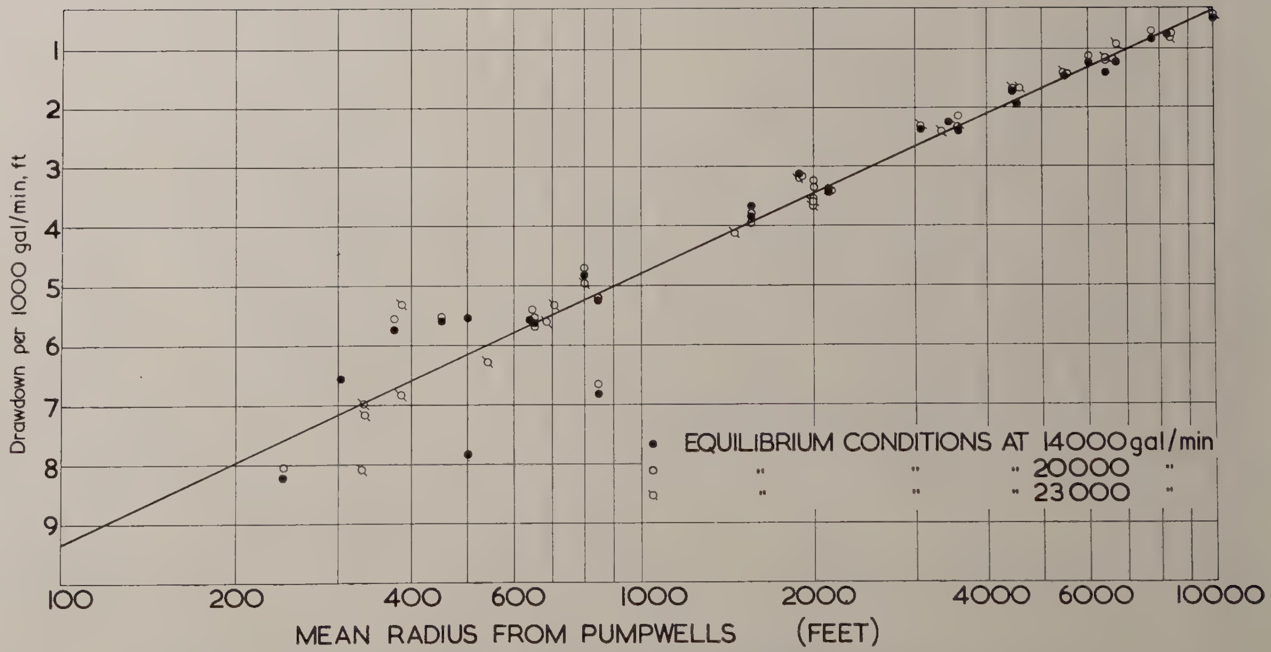


Fig. 11.—Variation of drawdown with distance per 1000 gal/min pumping rate (Neyveli).

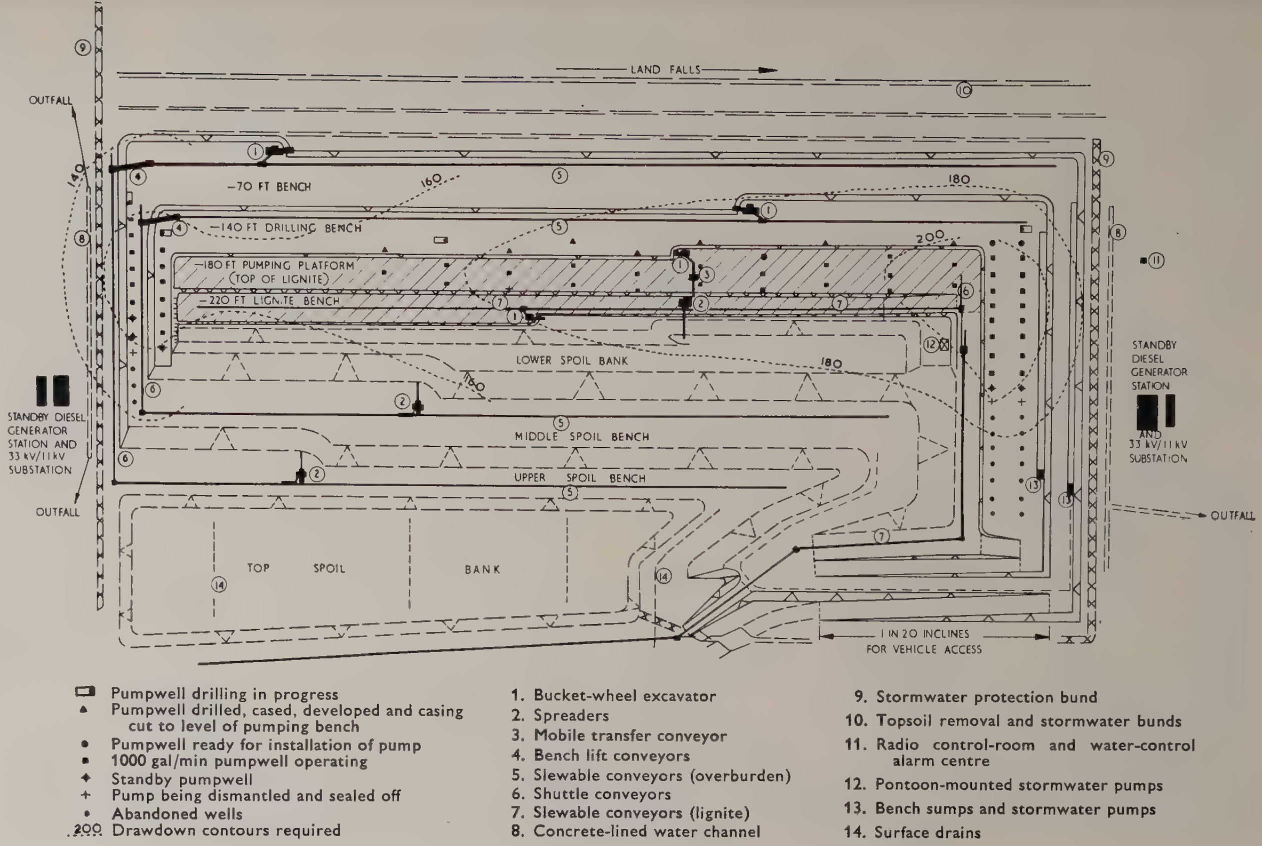

LAND FALLS
OUTFALL
STANDBY DIESEL GENERATOR STATION AND 33 kV/11 kV SUBSTATION
OUTFALL
STANDBY DIESEL GENERATOR STATION AND 33 kV/11 kV SUBSTATION
OUTFALL
−70 FT BENCH
−140 FT DRILLING BENCH
−180 FT PUMPING PLATFORM (TOP OF LIGNITE)
−220 FT LIGNITE BENCH
LOWER SPOIL BANK
MIDDLE SPOIL BENCH
UPPER SPOIL BENCH
TOP SPOIL BANK
1 IN 20 INCLINES FOR VEHICLE ACCESS
140
160
180
200
180
160
① ② ③ ④ ⑤ ⑥ ⑦ ⑧ ⑨ ⑩ ⑪ ⑫ ⑬ ⑭
▭ Pumpwell drilling in progress
▲ Pumpwell drilled, cased, developed and casing cut to level of pumping bench
● Pumpwell ready for installation of pump
■ 1000 gal/min pumpwell operating
✦ Standby pumpwell
+ Pump being dismantled and sealed off
• Abandoned wells
.200. Drawdown contours required
1. Bucket-wheel excavator
2. Spreaders
3. Mobile transfer conveyor
4. Bench lift conveyors
5. Slewable conveyors (overburden)
6. Shuttle conveyors
7. Slewable conveyors (lignite)
8. Concrete-lined water channel
9. Stormwater protection bund
10. Topsoil removal and stormwater bunds
11. Radio control-room and water-control alarm centre
12. Pontoon-mounted stormwater pumps
13. Bench sumps and stormwater pumps
14. Surface drains

Fig. 12.—Groundwater control scheme in the Neyveli lignite mine.

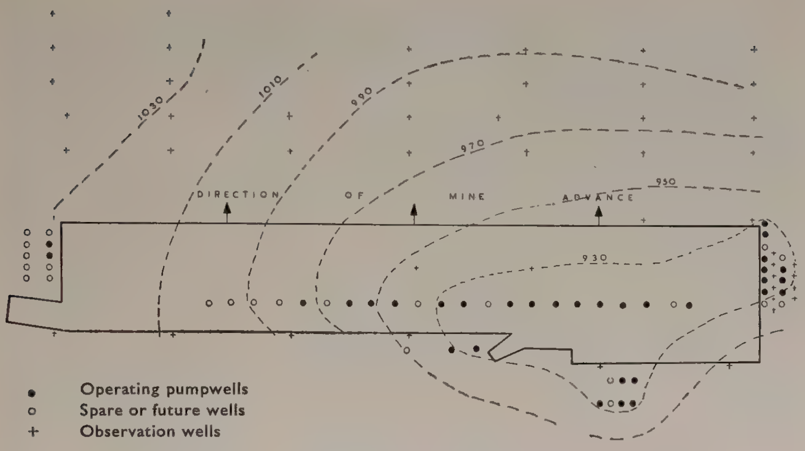
operation of a pumpwell of any capacity from zero to, say, 2000 gal/min at any other point in the region. The drawdown for 1000 gal/min is read off from the curve for the radial distance as taken off the plan and adjusted in proportion to the actual pumping rate. Using the principle of superposition, the influence at the same point of a second pumpwell, situated at any other location in the region, can be added and so on for other pumpwells until the required drawdown at the particular point is achieved.

By trial and error the optimum pumpwell pattern to achieve the required drawdown under the floor of the mine at all points in the most economical and convenient way is derived. This involves compromise as there are other considerations involved in a practical mining operation. Some desirable locations are not available for pumpwells, and others cause too much interference with other operations. Ideally the pumpwells would be placed outside the mine, but this invariably involves more pumping capacity and also excess drawdown, both of which are costly. In the case of Neyveli this would very likely have made the whole project uneconomic and would have been difficult to achieve because of the close proximity of the aquifer to the floor of the mine and the deeper local well depressions.

It is evident that the floor of the mine, i.e. the area to be protected, is a rectangle. The two parallel lines of pumps used in the pumping test were a first approximation to this shape, as well as being convenient in practice. However, two lines of pumps, equally spaced, produce elliptically shaped drawdown contours with a 'low' in the centre. This low can be flattened out if the pumps are concentrated at the ends, and the convenience of the end-bench locations finally resulted in the majority of the pumps being placed there. Nevertheless, hydrological and economic considerations still required some pumps to be situated on the working benches. The final pattern is illustrated in Fig. 12.

During the development of the mine the final drawdown pattern has to be achieved progressively as the box-cut deepens. Fig. 13 shows the position at Neyveli as at May 1963. The positions of the bore-holes have been selected for economy and convenience and the production-pit pattern can only be achieved progressively as the appropriate bench locations become available. This stage is the worst to be expected as the pump locations are not the most favourable and the benefit of long-term pumping will not have had effect.

The drawdowns required and achieved are indicated in the cross-section, Fig. 14. Line A–A indicates the equilibrium position where the weight of ground remaining above the aquifer just balances the hydrostatic pressure, i.e. there is no danger of heave until the excavation goes below this line (and even then the danger is small as a considerable thickness of strata still remains). However, pumps must be available



● Operating pumpwells
○ Spare or future wells
+ Observation wells

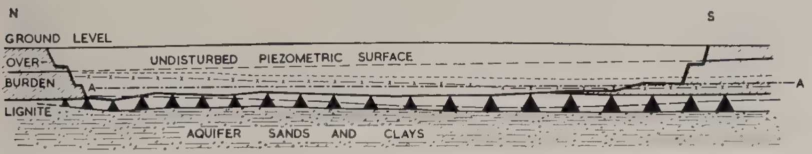
-950- Contours of piestic surface with continuous pumping at 34 000 gal/min (datum 1000 ft below mean sea level)

Fig. 13.—Development pumping at Neyveli (May 1963).

for continuous pumping at short notice to lower the piestic surface below the floor of the mine at all points as a protection against ingress of groundwater through old bore-holes, which was anticipated and did, in fact, occur at Neyveli.

Water disposal is by 16-in mild-steel pipes, each taking the discharge of four pumps. These run along the working benches and up the end-benches to discharge into open channels at the surface. The total planned discharge was 48 000 gal/min but, fortunately for the economics of the project, a large proportion of this water was required by the processing plants and the township, and the remainder was valuable for irrigation.

The power requirement of the pumps is of the order of 7500 kW, but again a 250-MW power station is situated close to the mine and this



Stages of drawdown attainable during standby stages of water control to protect mine against ingress of water through old bore-holes, etc.

First stage of continuous pumping for working below the heave line, pumping rate 34 000 gal/min (May 1963)

A——A Danger of heave for excavation below this limit

Fig. 14.—Longitudinal section of Neyveli mine showing stages of drawdown achieved in pumping to safeguard box-cut.

power is obtainable at low cost (and, incidentally, at virtually 100 per cent load factor). Six 1000-kW standby diesel generators were installed to provide emergency power supplies for 80 per cent of the pumps, particularly in the development period. Automatic-start features were adopted, both for the generators and the pumps, in order to restore pumping rapidly in the event of a power failure. This was considered essential in view of the very rapid rate of recovery of the water *pressure* (100 ft of head in the first hour) if the pumps stopped. In the case of actual dewatering the rate of recovery is much slower. This factor does not help in the case described as no dewatering of the main aquifer takes place, i.e. the inflow is always equal to the pumping rate.

The advance of the mine, which will average about 9 ft per week at a production rate of 3 500 000 tons per annum, requires the continuous drilling of new pumpwells (and observation holes) and the withdrawal and plugging of others as the advancing spoil heap threatens to cover them. In these circumstances the average life of a pumpwell, in the pattern shown in Fig. 12, is about 94 weeks and the minimum life 36 weeks, and this requires pumpwells to be drilled at the rate of about one per week.

Fig. 6 shows the sequence of drilling and developing, operation and abandonment of the pumpwell on the working benches. Drilling on the middle bench is necessary because a pit has to be made to accommodate the cuttings which cannot be permitted to contaminate the lignite. After completion and development of the pumpwell it must be sealed just below the top of the lignite seam and the casing cut off and removed above that point. This is done by dropping down a fabricated steel plug which comes to rest on a slight taper which was formed beforehand in the casing at this point. A casing-cutter is then used to permit the upper portion to be removed and thereby allow free passage of the bottom overburden excavator. The hole is then back-filled for safety.

When the overburden has been removed, the hole is re-located, opened up and a pump installed. Finally, the pumpwell is withdrawn from service and sealed before the lignite excavator reaches it. In case of difficulty, however, it could be kept in operation longer by repeating the cutting-down process, but it must finally be abandoned and sealed off before it is covered by the advancing spoil heap. If this were not done it would form a serious leak in the upper confining bed and permit water to rise under the spoil bank with the possibility of trouble later.

The total cost of pumping was about 6d. per 1000 gal, of which 50 per cent was power at a cost of about 1d./kWh. The amount of water pumped is estimated at about 32 tons per ton of lignite produced at the initial rate of output and about 19 tons at the final rate. The gross cost was expected to fall from about 4s. 6d. per ton to 2s. 6d. per ton of lignite and some credit was available from the sale of water.

In this mine pontoon-mounted sump pumps of a total capacity of 8000 gal/min are also provided to cope with monsoon rains and these provide an additional safeguard against flooding of the mine by groundwater. It must be realized that, in the event of failure of the pumps for more than a few minutes, the lowermost bore-holes would become artesian as soon as the piestic surface rose above the level of that bench. In that event, the space between the lignite face and the toe of the spoil heap would form a reserve sump and drain to the main sump pumps.

Acknowledgements.—Acknowledgement is due to the author's colleagues for their valuable help and encouragement, to the Neyveli Lignite Corporation and to Powell Duffryn Technical Services, Ltd., for permission to publish this paper. The opinions expressed are those of the author.

REFERENCES

1. Brealey, S. C. The Neyveli lignite project in India. *Min. elect. Engr*, **44,** Sept. 1963, 71–82.

2. Terzaghi, K., and Peck, R. B. *Soil mechanics in engineering practice.* (New York: John Wiley & Sons; London: Chapman & Hall, 1948), 566p.

3. Edmond, T. W. Some aspects of iron ore quarrying in Great Britain. *Trans. Instn Min. Engrs*, **122,** Aug. 1963, 806–25.

4. Todd, D. K. *Ground water hydrology.* (New York: John Wiley & Sons; London: Chapman & Hall, 1959), 336p.

5. Wenzel, L. K. The Thiem method for determining permeability of water-bearing materials and its application to the determination of specific yield results of investigations in the Platte River valley, Nebr. *Wat.-Supp. Pap. U.S. geol. Surv.* 679(*a*), 1936, 57p.

6. Theis, C. V. The relation between the lowering of the piezometric surface and the rate and duration of discharge of a well using ground water storage. *Trans. Amer. geophys. Un.*, **16,** 1935, 519–24.

7. Wenzel, L. K. Methods for determining permeability of water-bearing materials, with special reference to discharging-well methods. *Wat.-Supp. Pap. U.S. geol. Surv.* 887, 1942, 192p.

8. Cooper, H. H., and Jacob, C. E. A generalised graphical method for evaluating formation constants and summarising well-field history. *Trans. Amer. geophys. Un.*, **27,** 1946, 526–34.

9. Jacob, C. E. Radial flow in a leaky artesian aquifer. *Trans. Amer. geophys. Un.*, **27,** 1946, 198–205.

65.015:622.271.3:622.341.1(410) *Paper 19*

Two Examples of Work Study in Opencast Ironstone Mining

A. E. BALL

J. H. DANIELLS, A.M.Inst.W.S.

Work Study Officers, Ore Mining Branch of The United Steel Companies, Ltd.

SYNOPSIS

Two case histories have been selected from work carried out by the work study department of the Ore Mining Branch of the United Steel Companies, Ltd.

The first is an investigation into the working of a quarry using 30-ton capacity rear-dump trucks. The method of calculating the number of trucks required, using standard data obtained from previous studies and truck manufacturers, is described. The method of truck operation is then outlined, followed by a statement on cost.

The second is a study to assess the productivity of the locomotive fleet operating at an opencast ironstone quarry. There is a description of the method of observation, with an explanation of the results obtained. After a summary of recommendations made to management to reduce the number of locomotives by revised methods, there is a note on observations of work study officers after the introduction of the new methods, and an analysis of results.

Work study as practised in the opencast mining operations of a British steel company has two main functions. One is to examine existing work situations with a view to method improvements in the use of manpower and machines, the ultimate purpose being to increase productivity, expressed in terms of output per man shift or man hour, and to pursue a continual policy of cost reduction and safer working. The second function is to examine new schemes under consideration by management in order to establish new working methods, calculate the number of machines and men required and to estimate the operating cost of such new schemes. The results obtained and the proposals made by the work study department are discussed in detail and agreed with the manager concerned and subsequently presented in a formal report.

This paper describes two exercises, one in each of the above classifications, carried out by the work study department at two of the United Steel Companies' opencast mines in the Northamptonshire Sand Ironstone field.

AN INVESTIGATION INTO THE WORKING OF A QUARRY USING 30-TON CAPACITY REAR-DUMP TRUCKS

The Problem

Under the Ore Mining Branch's scheme for extending its iron ore producing capability it had been decided, at the appropriate time, to open up a new opencast mine in South Lincolnshire adjacent to an existing quarry.

The management had decided to use dump-truck haulage for the new scheme and a concrete trucking road would be used from the loading point at the quarry through to a loading dock at the existing railhead at the existing quarry.

The problem was to investigate the feasibility of using dump-truck haulage at the existing quarry in place of the steam locomotive rail haulage system so as to take advantage of the trucking facilities at the quarry which would be installed as part of the new quarry scheme. The fully developed scheme, which envisaged a fully gravity-operated marshalling sidings serving the loading dock, would preclude the use of rail haulage from the existing quarry face, and this provided an added reason for the investigation.

The specific questions posed were: at an output of 10 000 tons in 5 shifts—(*a*) the number of 30-ton dump trucks required; (*b*) the method of operating these trucks; and (*c*) the cost of operating the trucks.

Approach to the Problem

The first step, in common with the approach to all problems put to work study, was to gather together all the relevant data. These would then be examined in great detail and used to build up the answers to the questions posed.

In this exercise data regarding the geography and geology of the existing quarry, the line of quarry advance, the position of the concrete trucking road and loading dock were all well known. The operating data required on the 30-ton dump trucks were obtained partly from the vehicle makers' specifications and partly by synthesis (i.e. building up from elemental data) from studies of a fleet of smaller dump trucks owned and operated by the company on similar work at one of the other quarries in the area. Cost data were obtained from current price lists and the company's plant ledger and cost departments.

Determining the Number of Dump Trucks Required and the Operating Method

Trucks cycle

First the truck cycle time per load, and how this would vary as the loading point moved along the quarry face, had to be calculated. To do this it was necessary to consider the elements (i.e. small identifiable parts) of work constituting a truck cycle and to fit times to these. Manoeuvring and other operating times from previous studies of a similar operation were used as a guide. Truck loading times were calculated from a knowledge of the loading machine capability, the density of loose ironstone and the truck capacity. The 30-ton truck travelling times were based on the manufacturers' data sheets, taking into account the road surfaces to be travelled (concrete road, ironstone top and the ramps) with the required 25-ton load.

The availability of the trucks, that is the amount of time each truck would be likely to be available for trucking ironstone, taking into account starting up, shutting down, maintenance, etc., was based on previous study data, and gave a total of 2030 minutes per 42-hour week.

A typical cycle time is shown below.

	Min
Time to load average wagon 25 tons by $5\frac{1}{2}$-yd^3 dragline	3·00
Run loaded on ironstone top (say 300 yd at 5 m.p.h.)	2·04
Run loaded on concrete road to tipping dock (1600 yd at 18 m.p.h.)	3·03
Manoeuvre at tipping dock	0·33
Tip	0·50
Run empty on concrete road (1600 yd at 20 m.p.h.)	2·73
Run empty to loading machine on ironstone top (300 yd at 10 m.p.h.)	1·02
Manoeuvre at loading machine	1·25
Total cycle	13·90

Calculating the 'break-even' point

It can be seen from Fig. 1 that the cycle times would vary depending on the position of the loading machine. To allow for this a cycle time was calculated for each 100-yd marker along the quarry face. It was obvious that, as access could be obtained on to the ironstone top from either the southeast or northwest corner of the quarry, at some point along the face it would be as quick to run on from one end as the other. Owing to the different running speeds, loaded and empty, of the trucks on the concrete road and ironstone top, this point would not necessarily be halfway along the face. Subsequent calculation showed that this point, called the break-even point, would be about 850 yd from the southeast corner of the quarry.

Number of truck journeys required

The British Railways standard iron-ore tippler wagons of 27-ton nominal capacity, into which the ironstone would be loaded, hold on

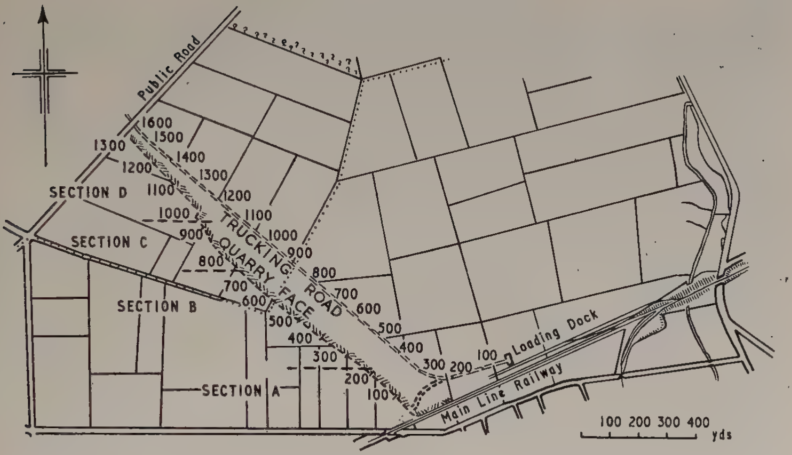
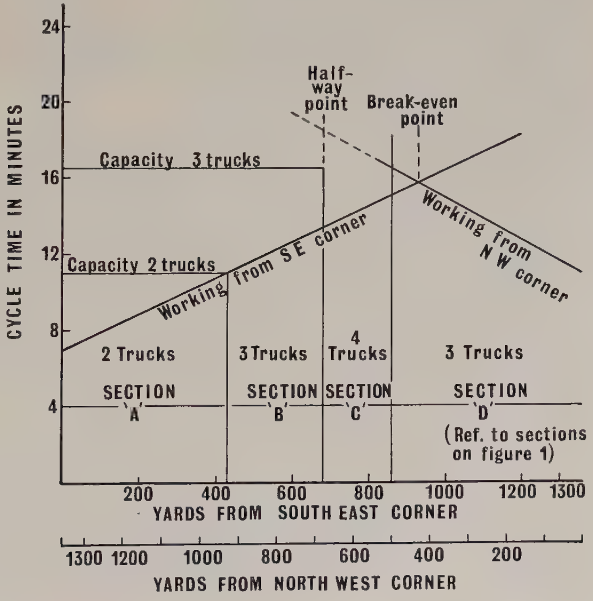


Fig. 1.—Plan of trucking road and quarry face.

average 25 tons of this ore; for an output of 10 000 tons per week 400 wagons would therefore have to be loaded.

It was considered that one dumper load was equivalent to one wagon load, so the problem was to calculate how many trucks would be needed, working a 42-hour week, to handle this tonnage.

Fig. 2.—Chart indicating no. of trucks required to produce 10 000 tons per week from existing quarry.



As mentioned previously, the cycle time for a truck load from each 100-yd marker along the quarry face had been calculated and the availability of the trucks was known. From this it was easy to say how many loads each truck could handle in the working week of 42 hours. These data were then used to produce Fig. 2, which shows, from the southeast corner and from the northwest corner, the relationship between the number of trucks required to produce 10 000 tons and the loading point expressed in yards from either corner.

The number of trucks required never exceeds four, so it would be necessary to have four trucks, but these would not be required to work full time. In fact the full face of the quarry would be worked along every 20 weeks at the stated output and, of this, four-truck operation would be needed for only 2 weeks. For the majority of the time, 11½ weeks, three trucks would be required, while two would be sufficient for the remaining 6½ weeks. This leaves a handsome margin of non-utilization of the four-truck fleet which could cover maintenance requirements and provide truck time for other work within the quarry group.

Method of quarry operation

In determining the method of truck operation some thought had to be given to the method of quarry operation as a whole. Briefly, the operation would consist of a 17-yd^3 shovel stripping off the limestone overburden (present average depth 31 ft) after it had been drilled and blasted by AN–FO explosive. The ironstone would be loaded by a 5½-yd^3 dragline standing on the ironstone top.

Figs. 3 and 4 show the method that was proposed. This takes account of safe working practice in that the trucks do not have to pass the stripping shovel while it is operating.

The loading machine would set in at the halfway point and work down the face to the southeast corner, taking out a 50-ft benching of ironstone and leaving the 30-ft of trucking road. This road would also be used by the loading machine to travel back to the halfway point to begin loading out to the northwest corner of the quarry.

While the loading machine was working to the southeast corner, the stripping machine would be stripping a cut from the centre portion to the northwest corner. It would then have sufficient width (80 ft) to travel back to the centre portion where the two machines would pass each other to begin the procedure in reverse.

Calculating the Operating Cost of the Four 30-ton Trucks

For this part of the study the help of the company's cost and plant ledger departments was enlisted. This close liaison is a regular practice within the company and ensures, as far as possible, that work study

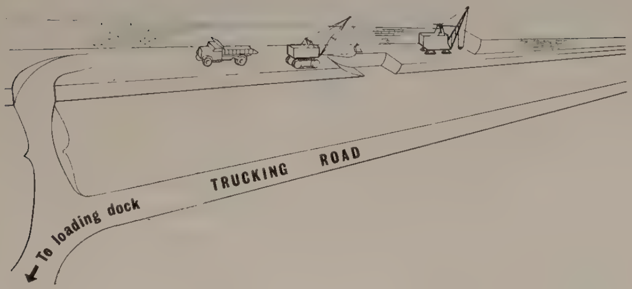


Fig. 3.—Method of working from southeast corner for existing quarry.

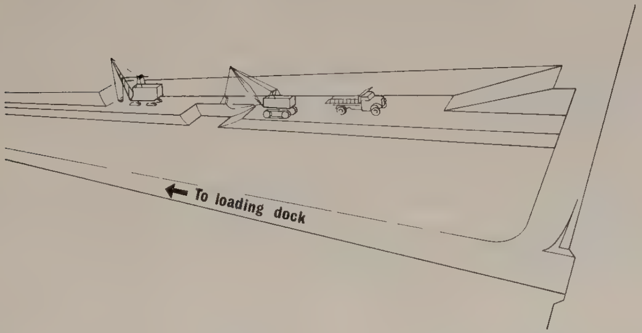


Fig. 4.—Method of working from northwest corner for existing quarry.

schemes are costed on a realistic basis. This means that any savings envisaged as the result of the introduction of a scheme should be achieved and management estimates put forward for sanction of expenditure can be based on the work study data provided.

The method of costing used was quite straightforward. The complete operation of the quarry was costed at the time, but the cost of truck operation only is shown here as being relevant to the present paper.

The annual operating cost for the four trucks was estimated as £26 088 and a breakdown of this figure is shown in Table I. Expressed as a cost per ton this works out at 1s. per ton at 10 000 tons per week over 52 weeks. It was pointed out, however, that this cost could be considerably reduced by more intensive operation of the quarry if higher tonnages were required, since the above figure only allows for the trucks working a 42-hour week. Expressed as a cost per ton-mile (this is perhaps a more comparative figure) the cost would vary, depending on the number of trucks in use along the various sections of the quarry face, as follows (assuming other work was found for the trucks not required):

Section (Fig. 1)	*Trucks*	*Pence per ton-mile*
A	2	11·3
B	3	9·3
C	4	8·7
D	3	4·5

It is interesting to note that the highest cost per ton-mile is along section A when the system is at its most efficient. The lowest cost per ton-mile, along section D, is the result of the higher running speeds obtainable on the concrete road and the longer haul.

TABLE I.—*Estimated cost of operating four 30-ton dump trucks*

	£ per year
Production wages (72·69d. per hour × 42 hr × 52 wk × 4)	2 646
Employment expenses, 14%	370
Depreciation (£40 per wk × 4 × 52)	8 320
Tyre replacement at one new set per truck every 10 months, cost per set £2060	9 870
Major engine and transmission replacements at total of £3000 per 5 years per truck	2 400
Miscellaneous spares at £300 per truck per year	1 200
Repair wages and oncost (£3 per wk × 52 × 4)	624
Fuel and oil at 2/6d. per operating hour	658
	£26 088

STUDY TO ASSESS THE PRODUCTIVITY OF THE LOCOMOTIVE FLEET OPERATING AT AN OPENCAST IRONSTONE QUARRY

This particular ironstone quarry was opened in 1952 and under planning permission the whole area must be restored to agriculture after working.

The quarry was laid out for railway transport in such a manner that there is an entrance at both ends of the working face and traffic circulates past the loading machine, thus minimizing delays while sets of wagons are changed. In other words, empty wagons are fed into the loading machine at one end of the system and loaded wagons go out of the quarry at the other end.

Fig. 5 shows the layout of the quarry with the sidings facilities at the weighbridge and exchange sidings for British Railways.

Most of the ironstone bed is drilled and blasted before loading. The loading machine is an all-electric face shovel excavator with a $3\frac{1}{2}$-yd^3 capacity bucket.

Management asked the work study department to carry out a study

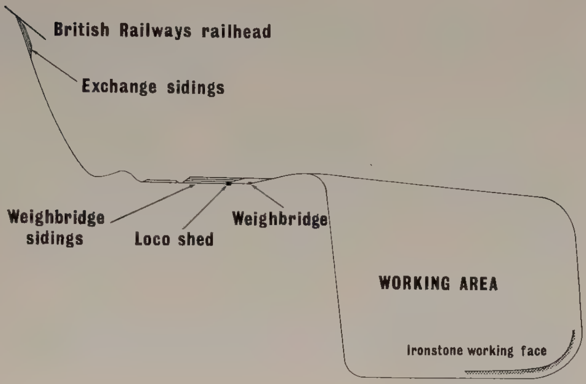


Fig. 5.—Sketch-plan of loco haulage quarry layout.

into the activities of the locomotives working this quarry and to provide an answer to the question: 'Is the existing locomotive service efficient?'

Description of Method of Observation

When the study took place five locomotives were used in the quarry, hauling empty wagons to the loading point and pulling loaded 27-ton iron ore tippler wagons to the weighbridge sidings. Another locomotive was engaged on hauling loads from the weighbridge sidings to the exchange sidings to be made available for British Railways to marshal trains of approximately 31 loads for onward transport to the blast-furnaces.

All locomotives working at this quarry are either 18-in or 16-in (cylinder size) steam locomotives purchased between 1950 and 1955.

A survey of the whole operation was carried out before the day of the study and, as only two observers were available, it was decided that one observer should be stationed at the loading digger and the other at the weighbridge sidings. Both stop watches were synchronized before starting the study, and by this means exact running times between both observer points were obtained. The observer at the weighbridge area was able to fix the time of each locomotive leaving with a set of empty wagons, and arriving back with loaded wagons. The observer at the loading point was able to determine when each locomotive arrived at the loading machine, the loading time for each set of wagons, and the time of departure loaded for the weighbridge area.

The observer in the weighbridge area was also able to watch the sixth locomotive, referred to in this study as the railhead loco, taking full wagons to the exchange sidings and returning with empty wagons.

From leaving the weighbridge area with empty wagons to returning with loads each loco travelled a distance of approximately 8 miles. The distance from the weighbridge area to the reception sidings travelled by the railhead loco was about 3 miles.

Owing to the distance involved it has been found necessary to arrange that three of the five locomotives in circuit working be available for work by 6.30 a.m. each day. They then collect their set of empty wagons (6 wagons for a 16-in loco and 7 wagons for 18-in loco) and travel to the loading point, to be available for loading starting at 7 a.m. The remaining two locomotives are available for work at 7 a.m. and they are at the loading point before the third set has completed loading.

When loads arrive in the weighbridge sidings they are weighed, labelled and placed in a 'full' road ready for the railhead loco to haul to the exchange sidings.

Data Obtained from the Study

Locomotive occupation time

Table II shows an analysis of the occupation time of five locomotives used in circuit working. From this it can be seen that they were doing effective work for approximately 53 per cent and were delayed or obstructed for 47 per cent of their total working time, the significant figures in Table II being alongside 'Waiting at digger to load'. This total figure of 432 min at once pointed out that, as sets of wagons were 'bunching up' at the loading point waiting to be loaded, the probability was that too many locomotives were being used. It was, in fact, ascertained that the shortest time a locomotive had to wait at the loading digger for the previous set to be loaded was 14·9 min, the longest time being 59·4 min.

Cycle time

The typical cycle time for each set of wagons was as follows:

	Min
Running to digger with set of empty wagons	30
Waiting at loading point for previous set to be loaded	27
Attending digger while set is loaded	24
Running to weighbridge with loaded set	32
Weighing and labelling of wagons	6
	119

In addition to this figure the servicing time for the locomotive was to be added: this covers taking on coal, water and oiling round, and amounts to 80–85 min per shift.

TABLE II.—*Analysis of occupation time for five locomotives*

Detail	Loco no. 10		Loco no. 12		Loco no. 36		Loco no. 39		Loco no. 13	
	Min	%	Min	%	Min	%	Min	%	Min	%
Running with empty wagons	113·4	19·4	86·0	15·1	87·8	16·3	94·0	18·4	95·9	17·8
Running with loaded wagons	126·1	21·6	153·6	26·9	98·3	18·2	94·1	18·5	99·9	18·5
Weighing	9·5	1·6	5·8	1·0	4·9	0·9	4·7	0·9	6·8	1·3
Attending digger loading	96·3	16·5	89·4	15·7	59·3	11·0	58·3	11·4	77·4	14·3
Total work	345·3	59·1	334·8	58·7	250·3	46·4	251·1	49·2	280·0	51·9
Waiting at digger to load	73·7	12·6	65·9	11·6	87·2	16·1	100·1	19·6	105·3	19·5
Traffic delays			10·0	1·7	50·3	9·4	2.7	0·5	26·7	4·9
Waiting for labelling of wagons	14·4	2·5	11·4	2·0	10·6	1·9	10·0	2·0	10·7	2·0
Shed time, including servicing on job	151·6	25·8	147·9	26·0	141·6	26·2	146·1	28·7	117·3	21·7
Total delays	239·7	40·9	235·2	41·3	289·7	53·6	258·9	50·8	260·0	48·1
Total time	585·0	100·0	570·0	100·0	540·0	100·0	510·0	100·0	540·0	100·0
Number of sets	4		3		3		3		3	
Wagons per set	7		7		6		6		6	
Total wagons	28		21		18		18		18	

TABLE III.—*Analysis of occupation time for four locomotives*

Detail	Pit locos							
	Loco no. 10		Loco no. 12		Loco no. 13		Loco no. 11	
	Min	%	Min	%	Min	%	Min	%
Running with empty wagons to digger	92·0	15·7	100·0	16·3	100·0	16·6	77·0	14·2
Running with full wagons from digger	141·2	24·2	129·2	21·0	128·2	21·4	114·9	21·3
Attending digger while loading	117·0	20·0	129·0	21·0	120·0	20·0	88·0	16·3
Weighing	8·4	1·4	8·4	1·3	8·4	1·4	6·3	1·2
Other work							26·0	4·8
Running with loads, returning with empties from exchange sidings								
Total work	358·6	61·3	366·6	59·6	356·6	59·4	312·2	57·8
Wait for previous set to be loaded	49·0	8·4	60·0	9·8	84·0	14·0	54·0	10·0
Wait for labelling wagons	14·4	2·5	14·4	2·3	14·4	2·4	10·8	2·0
Shed time and servicing, coal and water	163·0	27·8	174·0	28·3	145·0	24·2	163·0	30·2
Total delays	266·4	38·7	248·8	40·4	243·4	40·6	227·8	42·2
Total time	585·0	100·0	615·0	100·0	600·0	100·0	540·0	100·0
Number of sets	4		4		4		3	
Wagons per set	7		7		6 (1) and 7 (3)		6 (1) and 7 (2)	
Total wagons	28		28		27		20	

Loading digger operating times

The digger loading time per set of seven 27-ton wagons averaged 26·5 min, and for sets of six wagons averaged 21·7 min.

The actual loading time of the loading digger was 79·3 per cent of an 8-hour shift, the balance of 20·7 per cent being small delays between each set loaded, due to spillage on the rail track, and servicing time. The servicing time covers inspection of machine before beginning work, greasing and oiling, and minor adjustments to machinery.

On the observed day this digger loaded 2696 tons of ironstone in 7 hours 8 min total time, an average of 378 tons per working hour.

Recommendations

All details of the study were reported to management and it was suggested that a proposed new method of working be considered, owing to the long delays of locomotives standing waiting to load behind the loading digger. It was proposed that as this delay amounted to 432 min per shift it was possible that the locomotives working this quarry could be reduced from five to four, and still maintain the rate of output required.

Management agreed to this proposal, and a further study of the new method of working took place a few days after the scheme was introduced.

Data Obtained from New Method of Working with only Four Locomotives in Circuit

With this new method of working two work study observers were stationed at the same points as in the original study in order to obtain comparable data.

Locomotive occupation time

Table III shows a full analysis of the occupation time of the four pit locomotives, plus the railhead loco. It can be seen that the effective work of four locomotives has now been increased from 53 to 60 per cent and delays reduced from 47 to 40 per cent against the old method of working.

Again the significant figure is alongside ‘Wait [at digger] for previous set to be loaded’, the total being 247 min, against 432 min on the old method of working—a reduction of 43 per cent.

Loading digger operating times

The actual loading time of the digger was 91·7 per cent of an $8\frac{1}{4}$-hour shift, the balance of 8·3 per cent being small delays between each set loaded due to cleaning up spillage and setting wagons.

On the observed day the digger loaded 2624 tons of ironstone in 7 hours 34 min total loading time, an average of 347 tons per working hour.

Results of Studies of New Methods

The operation of the new method of working proved very satisfactory, resulting in a higher utilization of all locomotives engaged in hauling ironstone from the quarry. The reduction of one locomotive and crew is estimated to be saving the company £4200 per annum.

Discussion

(Papers 16–19)

Mr. R. Glossop,* introducing Papers 16 to 19, said that their range was such that it was extremely difficult to generalize about them. Fortunately, they fell into two fairly well defined classes. Papers 16 and 19 were examples of new ideas in management and, if engineers were to retain and increase their place and influence in society, they must take management very seriously indeed. Papers 17 and 18 were examples of the contemporary approach to engineering geology—an important topic.

Dealing with Paper 16, 'Use of a computer for grading iron ore at Sept-Iles, Quebec', it might be said that computers had already shown their value in project planning, and particularly in network analysis for large and complicated operations. A large computer with all its associated equipment, with a suitable building to house it, might cost something in the order of £500 000. Most people therefore for the present had to be content with buying time, which fortunately could be done and, where large projects were concerned, was well worth doing.

A modern digital computer, such as the AEI 1010, when dealing with critical path analysis, would actually produce and print a bar diagram, and since few complex operations could ever be carried out strictly to programme, that was an immense advantage for on occasions it was a simple matter to amend the programme, allowing for any failures to maintain progress, and from the computer, in a very short time, to obtain a new critical path analysis, and a revised bar diagram as a guide to future work, allowance being made for the preceding failure to keep to programme.

The problem for most firms, he imagined, was that of getting a sufficiently high utilization from a computer, even though it would deal with such matters as pay sheets, staff salaries, costs, estimates, stock control and network analysis.

In the case described by Dr. Kirkland and Mr. Waring the principal use of the computer was in grading and blending iron ore, though presumably it must be used for other purposes. In that instance it was striking that although two years were required to investigate the application of a computer to the problem and for programming the system and although there had been no financial saving, it had been considered well worth while to install a computer for other reasons. For example, because the actual process of blocking and blending was carried out more effectively than when done manually and, perhaps more important, that data not previously available were now under control. It would be interesting to know if the increase in efficiency of those calculations had led to a re-examination of the method of ore analysis and sampling.

---

*Soil Mechanics, Ltd., London.

'Two examples of work study in opencast ironstone mining' by Mr. Ball and Mr. Daniells, dealt with work study—one more example of a tool of management in engineering. In fact, most of that which the work-study engineer now did, every engineer had done, or should have done, in the past, albeit less effectively. The work-study engineer had the advantage of being trained to analyse complex operations into their elements of work, and to relate the actual rate of working of labour to a recognized standard rate of performance. Proper allowance was also made for relaxation and conditions of work. The resulting standard time for an operation was of value to management in determining the best methods of working.

Nevertheless, it was not always the case that the employment of a specially trained engineer was possible, and he thought that those with handling problems of that type would learn much from the paper, which showed what could be achieved by a logical and scientific approach.

There was one feature of the paper which should be mentioned; it was essentially an analysis of the use of plant—not an investigation into the efficiency of labour.

Papers 17 and 18 were examples of the 'new look' in engineering geology, by which he meant the application of quantitative methods to engineering problems which involved geological materials. The term 'engineering geology' (and in that he did not include mining geology or economic geology) had long existed but, in fact, until very recently no corresponding subject existed although, of course, competent field geologists had often given advice of the greatest value on problems of constructional engineering. If that appeared too sweeping a statement he thought it would be justified if all the textbooks on engineering geology published up to about 1938 were examined. They would be found to be little more than standard works on elementary geology with a few, not always very well chosen, examples of its application to engineering.

The quantitative approach to geology started as long ago as the middle of the eighteenth century, when French military engineers concerned with fortifications studied the pressure of earth on retaining walls, but progress was very slow. In 1846 Collin published a valuable analysis of the stability of clay slopes, but it was ignored for the better part of the century.* About the same time d'Arcy enunciated the law describing the flow of water in sand.† Other contributions of varying importance were made from time to time during the second part of the nineteenth century, but it was not until 1925 that Terzaghi's *Erdbaumechanik* appeared,‡ and it was there that he published the definition of effective stress $\sigma' = \sigma - u$, a deceptively simple statement which was the foundation of soil mechanics. That set off a chain reaction, and from then on the science of soil mechanics grew rapidly, soon being accepted as an essential part of engineering science. During the last 25 years or so the problem of rock bursts in deep mines had led mining engineers to study the mechanics of rock masses and, more recently, civil engineers had extended the methods of soil mechanics, at first to very weak rocks, and later to all rocks. The necessity for that work had become evident following several very serious dam failures in recent years.

As matters now stood there was a large field of knowledge of those subjects which was bound to coalesce in the near future to form a separate discipline. There was a tendency to refer to this subject as 'geotechnology', but perhaps its real name was 'engineering geology'.

Mr. Meigh and Professor Henkel, in their paper 'Applications of rock and soil mechanics to surface mining', had undertaken the difficult task of providing a general paper treatment of the subject, on the assumption that most of their readers would have had no special training in the field. Their short and logical introduction to the subject would be a useful guide to mining engineers and geologists who had had no previous acquaintance with it and he hoped that it would lead them to further reading and study of that interesting branch of applied science.

At the beginning of their paper the authors pointed out that the study of the stability of slopes in soil and rock had immediate applications to opencast mining, and that, whereas in relatively permanent

---

*Collin, A. *Recherches expérimentales sur les glissements spontanés des terrains argileux* (Paris: Carilian-Goeury, 1846). Collin, A. *Landslides in clay* (Translation of 1846 publication by W. R. Schriever). (Toronto: University Press, 1956).

†d'Arcy. Les fontaines publiques de la ville de Dijon (Paris: Dalmont, 1856).

‡Terzaghi, K. *Erdbaumechanik auf bodenphysikalische Grundlage* (Leipzig and Vienna: F. Deuticke, 1925). References to this work, which is now rare, will be found in *From theory to practice in soil mechanics: selections from the writings of K. Terzaghi*, J. Wiley & Sons, 1960.

structures, such as cuttings for motorways, railways, etc., slopes must be made stable for very long periods, in the case of temporary excavations (for example deep foundations) and particularly in the case of the advancing faces of opencast mines, it was only the short-term stability which mattered.

A short section on soils went at once to the very heart of soil mechanics theory by pointing out that the essential difference between sands on the one hand, and clays on the other, depended, from the point of view of soil mechanics theory, upon the relative rates at which pore-water pressures were dissipated in those materials. They also distinguished between normally consolidated clays and over-consolidated clays.

The stability analysis of soil slopes was a tedious process, although in recent years the labour had been lightened by Bishop and Morgenstern's publication* and by the use of computers. There the authors wisely observed that where the soil profile was complex, theoretical analysis of that type might be valueless, and one must fall back on previous experience of similar conditions. That should not, of course, lead one to assume that the theoretical approach was not very valuable in many cases.

The authors then returned to the question of long-term and short-term stability problems, and stated that long-term stability, which fortunately could rarely be of much importance in opencast mining, could only be calculated by using the effective-stress shear parameters and pore-water pressures corresponding to the final equilibrium ground water conditions.

The short-term stability condition could, however, be calculated from a $\phi = 0$ analysis, using the undrained shear strength of the clay, which was easily measured. An interesting example of the failure of a slope in the London Clay showed that in fissured clays, although the final long-term stability condition could be calculated, comparatively little was yet known about the rate of change of stability with time; from the example quoted it would seem that in stiff fissured clays the decrease was at first very rapid.

Site investigation methods were dealt with briefly, with reference to soil sampling and to *in situ* measurements of permeability, and to the measurement of pore-water pressures by means of piezometers. A short section was devoted to the triaxial compression test, but since modern methods of the measurement of shear strength on soil samples were highly sophisticated, it did no more than explain that there were two principal kinds of triaxial test applicable to slope stability problems, the drained and the undrained, or that in which the rate of testing was so slow that excess pore-water pressures were allowed to dissipate during the test and effective stresses were measured directly; and the other where no pore-water was allowed to escape from the sample during testing.

---

*See reference 2 on page 389.

In considering rock slopes the authors pointed out the importance of the orientation and spacing of joint planes and of the water pressures within them, and also referred to the *in situ* stress in the rock. Much work was going on in the field of rock mechanics at the present time in devising instruments which would measure the principal stresses in a rock mass.

As regards methods of improving stability, it was shown that the most generally useful method, both for soils and rocks, was that of drainage, although in certain cases rock bolts and other forms of anchorage could be used successfully.

The theoretical introduction was followed by a short description of actual cases of slope stability problems taken from recent practice, and the paper concluded with the wise observation that although analytical procedures were of great value, account should always be taken of the overall geological environment.

'Groundwater control in opencast mining' was a particularly interesting paper; it was, he believed, the first account in English of the use of groundwater lowering by means of tube wells in mining.

The method of groundwater lowering by wells as an aid to deep excavation in water-bearing sands was discovered by Robert Stephenson in 1838 during the construction of the Kilsby tunnel on the London–Birmingham railway. Work was brought to a halt when the headings intersected a stratum of running sand within the Interior Oolite Series. Stephenson decided to drain the stratum by sinking wells at intervals along the line and slightly to one side of the axis of the tunnel. The wells were about three ft in diameter and were sunk by the conventional method of the day. They were equipped with pumps operated by Cornish engines and flat rods. 'As the pumping progressed', wrote Stephenson, 'the most careful measurements were taken of the level at which the water stood in the various shafts and bore-holes; and I was soon much surprised to find how slightly the depression of the water-level in the one shaft influenced that of the other, notwithstanding a free communication existed between them through the medium of the sand, which was very coarse and open. It then occurred to me that the resistance which the water encountered in its passage through the sand to the pumps would be accurately measured by the angle or inclination which the surface of the water assumed towards the pumps, and that it would be unnecessary to draw the whole of the water off from the quicksand, but to persevere in pumping only in the precise level of the tunnel, allowing the surface of the water flowing through the sand to assume that inclination which was due to its resistance'.*

Professor Boyd Dawkins in his James Forrest Lecture of 1898, described the result as 'a dry pathway for the railway between the waters heaped up at the angle of friction on either side, like those represented in some of the old Bibles in the passage of the Children of Israel through the waters of the Red Sea'.†

In spite of its obvious advantages, that process was little used until the development of electric submersible bore-hole pumps in Germany just after the First World War. Being very convenient for the purpose, groundwater lowering was thereafter widely used for deep excavations in the sandy plains of northern Germany and, in particular, on the construction of the Berlin Underground railway, much of which was done in open cut.

He would like to draw attention to several points mentioned by Mr. Brealey. First, a proper investigation at the planning stages might save much delay and expense later. That might sound obvious, but it was often neglected. Secondly, the strata should indeed be tested well below the base of the proposed excavation and, in cases of complex strata, exploratory bore-holes should be carried to a depth below bottoming-up level of 50 to 100 per cent of the total depth of excavation below the ground level.

With regard to sampling, he did not feel very happy that the sludge from reverse-circulation bore-holes should be the only source of formation samples. Those might be good enough once the geological structure and the geotechnical properties of the various beds were fairly well known over the area, but in the exploratory stage the location and sampling of trial bore-holes could not be too carefully done, and if that cost a little money it was nevertheless money well spent. Those exploratory holes could generally be sited so as to be used at a later stage as observation wells.

He agreed that sieving tests followed by the use of Hazen's formula gave only a rough indication of the permeability of an aquifer, and in most cases pumping tests were the only reliable guide. The difficulty was, of course, that the vertical permeability was often so very much less than the horizontal permeability, owing to the alternates of finer

---

*Stephenson, R. *Second report to the directors of the London, Westminster and Metropolitan Water Company*, 1841.

†Dawkins, B. On the relation of geology to engineering. *Proc. Instn Civil Engrs*, **134**, 1898, 254–77.

and coarser microstrata almost invariably present in sedimentary material. He had found that undisturbed samples of sandy beds taken with a compressed-air sampler were useful at that stage.

As regards the installation of wells, the author pointed out the value of reverse-circulation drilling and the use of submersible bore-hole pumps, stressing the need to exclude fine sand from the pumps, both by the proper use of filters and by a preliminary stage of over-pumping, using an airlift. If very fine quartz sand passed through the filter in any quantity its effect on the pumps could be disastrous and the cost of maintenance very high.

Mention of a standby diesel generator with automatic-start switches to restore pumping at once in the case of a power failure was important; it was indeed an essential feature of all large water-lowering installations.

The ideal place for the wells was outside the excavation, and they were invariably so placed in civil engineering works, but the Neyveli project was unusual and it would be of great interest to construction engineers to read of the water-lowering installation designed to deal with a high face advancing at a rate of 9 ft per week over a period of years. The author also mentioned that the life of some of the wells was up to 94 weeks. He would like to know whether any difficulty arising from the precipitation of iron salts in the filters in the wells had been encountered. That sometimes caused trouble in the London area where wells were in operation for more than a year.

Professor R. A. L. Black* said that in his excellent paper (p. 391) Mr. Brealey wrote: 'Where the overburden and ore are consolidated rocks there is no stability problem due to groundwater. The faces may be maintained at steep angles and even the rock dumps are not endangered. Although groundwater may be present in quantity, particularly if the rock is fissured or porous, it may be dealt with purely as a drainage and pumping problem without concern for stability failures'. That, of course, depended on what one meant by 'steep'. Slopes in quartz–porphyry could be stable at slopes of 70 to 80° over considerable heights and failures could occur at slope angles as low as 25°. It depended on the conditions, there being two principal factors involved:

(*a*) the structural controls—the faults, joint patterns and bedding planes; and

(*b*) the groundwater pressure exerted in fissures and joint planes.

---

*Royal School of Mines, London.

An accident that he was recently asked to investigate was almost certainly due to fissure water pressure acting on a natural joint plane behind a 20-ft vertical face in a chalk quarry, which led to an unexpected collapse at the face. The mechanics involved in the occurrence were not clearly understood by the contractor in charge of the operation with—in that case—fatal results.

The analytical methods described in Papers 17 and 18 were, in fact, equally applicable to slopes in soil-type materials and to slopes in hard rock. The only real difference was that in the latter case, in hard rock, they were very much more difficult to apply. Nevertheless, it was very important from the points of view of both economics and safety that the methods of scientific analysis be applied to slopes in rock in open-pit work as well as to soil slopes.

He was sure that Mr. Brealey would not disagree with his remarks. Reading between the lines of his paper, he suspected that the author's apparent dismissal of the need for a scientific approach to engineering slopes in hard rock was due really to the fact that he did not happen to have become involved in such problems himself.

Mr. E. Rich,* referring to Paper 17 by Mr. Meigh and Professor Henkel, said that on page 377 reference was made to the use of rock bolts to increase the stability of rock slopes. He would like to hear of examples of that application. He was particularly interested in knowing what increased slopes were obtained as a result of the bolting. It would also be very helpful if the authors could give some details as to the diameter and length of the bolts and particulars of the materials used for them.

Some years ago he had studied such a possibility in regard to the steepening of the overburden slopes in a large opencast operation—which procedure appeared economically attractive. The alternative of uncovering more ore by taking the benches back at the existing slopes would have been totally uneconomic.

Mr. J. C. Hood† said that reference was made (pp. 385–8) to a study of Masaboin Hill in Sierra Leone with the object of determining the maximum slope permissible when mining the decomposed quartz–mica schists, locally known as sterile. It was suggested that 'slopes of the order of 70° to the horizontal should be used, with a vertical interval between benches not exceeding 40 ft'. The authors went on to say (p. 388): 'These recommendations were related to a very long-term mining programme and it will be some years before the predictions can be related to field results'. Those tests were made in 1959 and some 5 years later mining had progressed to the stage where 110 RB shovels were being used for excavating vertical or near-vertical faces of approximately 30 ft in height in the quartz–mica schists in question. It might be of interest to state that, generally, those faces were standing, although in some cases slips had occurred from 10 to 20 ft back from the crest of the face, causing quite large-scale slumping of material for a vertical distance of 6 to 10 ft and, in one recent case, up to 20 ft.

---

*International Nickel, Ltd., London.

†Mining Engineer.

Mr. A. A. Archer* said that in Paper 17 the term 'soil', as used in civil engineering, was defined, but it was regrettable that that was still necessary, its meaning being unique to that particular discipline. That might be illustrated, for example, by the fact that sand and gravel were included in the author's definition, while operators of sand and gravel workings used the term 'soil' in the generally accepted sense. To some there was confusion between the roles of soil mechanics engineers and pedologists; the authors defined 'soil' as 'relatively weak and unconsolidated' material, but clays were classified as 'normally consolidated' or 'over-consolidated'. Just as the term 'geotechnics' was replacing that of 'soil mechanics', it was to be hoped that an alternative to 'soil' would soon be adopted.

Mr. H. B. S. Dales† said that he agreed with Mr. Glossop's comments on Mr. Brealey's paper on the necessity of supplementing information obtained from sludge samples from reverse-circulation bore-holes by samples obtained by cored bore-holes, and on the subsequent use of these cored holes as observation holes. He would also welcome the author's remarks on what he considered to be the minimum diameter for pumping-test bore-holes (*a*) in loosely consolidated sands and (*b*) in well consolidated but permeable beds.

Mr. Brealey's advice as to the methods to be employed if a virgin area to be excavated consisted of hard, but probably well fissured, limestone would be welcome. It was often of vital interest to know the amount of dewatering that would be required in a large quarry opened in such material. It would appear that, at present, the best that could be done was to determine firstly the most highly fissured areas and then to carry out pumping tests in those areas. That method, however, was by no means reliable and suggestions as to how further reliable evidence could be obtained would be of great value.

---

*Geological Survey of Great Britain, London.

†Position unknown.

Mr. V. H. R. Oliver* said that the paper by Mr. Meigh and Professor Henkel would be of great use to the practical mining engineer during his design of an open pit, with particular reference to the choice of a safe but economic pit slope. No mention was made, however, of the need to consider the location of spoil dumps in relation to the open pit.

Since, among other things, the stability of a pit slope was a function of its height, it was felt that the proximity of a surface waste dump to a pit slope was a factor for consideration in the overall design of opencast workings. If situated close to the pit the waste dump might artificially increase the height of the pit slope. Did the authors consider that an important factor and, if so, would they give details of the manner in which it should be treated during the design of a pit?

Mr. H. E. K. Allen† said that the authors of Paper 17 had shown that when the bedding of ground and general slip planes dipped towards the pit face unfavourable pit stability conditions could arise. As discussed during Session 3, greater explosive efficiencies were gained when a face could be orientated so that bedding planes dipped in that very same direction. Thus the engineer who designed pits had to decide on the advantages gained through safer (and possibly steeper) pit faces and those from decreased cost of explosives.

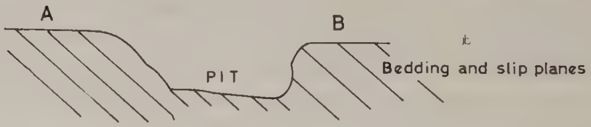


Fig. A.—Section showing alternative orientation of pit faces with respect to bedding. A, Favourable for blasting, but possibly dangerous if steep slope angles were required; B, Inefficient for blasting, but generally more stable and safer faces.

---

*Chambishi Mines, Ltd., Zambia.

†De Beers Consolidated Mines, Ltd., London.

WRITTEN CONTRIBUTIONS

Mr. D. S. Paterson:[*] Paper 17 refers to the importance of detailed knowledge of the local geology and a few comments on the sources of such information may be useful to mining engineers and others interested primarily in underdeveloped countries overseas.

The Mineral Resources Division of the Overseas Geological Surveys provides among its many services to the mining industry an information service on the organizations, references and maps concerned with the geology, mining, mineral and water resources of specific countries overseas, with emphasis on the underdeveloped ones. This service is based on the extensive records and indexes of the Division and is disseminated by means of 'Preliminary Notes' or by the provision of literature and for study in its reference library.

Mr. M. Mukai:[†] I would like to ask Mr. Meigh and Professor Henkel if there is any device or instrument that can be used in the field to foretell slope failure in soil and rock.

Mr. R. O. Morris:[‡] I wonder if Mr. Brealey has considered the possibility of freezing groundwater along the margins of opencast workings. That would mean that pumping would only be required to drain the actual working areas, it would eliminate the effects of groundwater over large surrounding areas and would produce firm and stable margins to working areas which would be safer and might result in narrower working areas.

Mr. E. Rawcliffe:[§] Mr. Meigh and Professor Henkel quote (pp. 372–3) two examples of slips in London Clay in Essex. Case *A*, an excavation 48·5 ft deep was stated as failing within 1 day of completion and case *B*, 45 ft deep, within 19 days.

Would the authors give the period over which these excavations were made, which was probably much longer than the failure time quoted?

---

[*] Overseas Geological Surveys (Mineral Resources Division), London.

[†] Mitsui Mining and Smelting Co., Ltd., Philippines.

[‡] L. M. Van Moppes (Diamond Tools), Ltd.

[§] W. and C. French, Ltd., Essex.

Excavations in over-consolidated fissured clays produce tensions and shears which open up the fissures, so that 'solid' clay offering full shear resistance occurs over only part of any particular slip plane. The shear resistance along that part of the slip plane passing through opened-fissured clay would be greatly reduced. Thus without actual softening of the clay taking place, the *average* shear resistance along the slip plane might be reduced to the equilibrium or less, and this would account for the rapid failure which occurs in some excavations in stiff-fissured clays, rather than by considerations of pore pressure. The authors' comments on this point would be useful.

Mr. K. A. Fern:* In reading Papers 17 and 18, and following the discussion at the symposium, I have been struck by the assumption (which appeared to be general among mining engineers, civil engineers and geologists) that where groundwater presents a problem the only solution is the installation of drainage wells and/or the use of an elaborate pumping system. I am especially interested in this problem as it occurs in deep mines and large dams, where the cost of constant pumping may be a significant item in operating charges. Engineers are therefore more and more considering the possibility of permanently sealing water-saturated strata or diverting flow to areas where they do not affect operations.

Chemical companies in Europe and America have developed a number of chemical grouts which can be injected into porous formations, forming completely impermeable water barriers. It is quite probable that some of these products might also be used to control ground which is liable to slip, though here the criterion will be the porosity of the strata and the ability to accept grout under pressure without causing sufficient 'heave' to disturb the slope. I believe some studies along these lines are currently being made at Neyveli.

Regarding the risk of damage to pumps by abrasive material, in a number of mines on the Witwatersrand water is pumped from great depth and, owing to the abrasive nature of the rock, maintenance was a very serious problem. By constructing large sumps ahead of the pump stations, and arranging a continuous feed of a polyelectrolyte to the sumps, the greater part of the abrasive quartz, etc., settles out and the feed to the pumps is virtually clear water. Similarly, the chemist can be of help in preventing the clogging of filters and a number of specialized sequestrants are available for this purpose.

---

*Cyanamid of Great Britain, Ltd., London.

Author's Replies

(Papers 16–19)

Mr. A. C. Meigh and Professor D. J. Henkel: In reply to Mr. E. Rich, (p. 436), the increase in slope that can be achieved economically depends on the nature of the rock, the joint pattern and the strength characteristics of the joint planes. The effectiveness of the rock bolts depends on the increase in the effective stress they induce and the possibility of creep reducing the tension in the bolts; thus the effective stresses must be considered.

Most of the literature on rock bolting of slopes relates to civil engineering practice rather than to mining, and in many cases it has been used to give support to a slope which cannot be flattened because of space limitations or for other reasons, rather than to achieve a steeper angle in slope design.

Diameters for bolts often follow tunnelling practice; long-life materials are of interest in many applications; length of bolt is a function of the geometry of the particular problem, but in general considerably larger bolts are required than those used in roof bolting.

We were very interested in Mr. J. C. Hood's information (pp. 436–7) concerning the performance to date of the excavation slopes at Masaboin Hill. The authors stated in their paper (pp. 387–8) that '. . . owing to the orientation of some of the bedding planes the use of vertical faces may lead to local slips and it was therefore suggested that, as a compromise, slopes of the order of 70° to the horizontal should be used'. Apparently, vertical or near vertical slopes had been used, and in some cases local slips had occurred, as predicted.

In answer to Mr. V. H. R. Oliver (p. 438), the location of spoil dumps in relation to the edge of the excavation is of great importance. There are a number of examples where the placing of spoil dumps too close to the edge of excavations has led to failures. The methods of stability calculation mentioned in the paper may be used to investigate the safe distance for the location of spoil dumps.

It is to be noted that in the example given in the paper of unstable excavations in London Clay in Essex (Fig. 1) fill material is involved in the unstable mass. This was, in fact, spoil from the excavations.

In answer to Mr. M. Mukai (p. 439), in many cases the failure of slopes in soil and rock are preceded by the development of tension cracks or by a gradual increase in the rate of deformation in a potential slide area. Regular readings of slope movements will often give warning of the development of slide conditions. The telemetering of cracks in rocks has been used in Norway to give advance warnings of rockfalls.

It is not possible to answer precisely Mr. E. Rawcliffe (pp. 439–40) about the periods over which the excavations were made. The slips occurred in parts of the perimeters of two large excavations, which were put down fairly quickly, working in sections. It is probable that for cross-section *A* the total excavation period was less than one week, and the earlier part of this period would not be significant because of the shallow depth. The significant period of excavation is probably only 3 or 4 days.

The strength of fissured clays in excavations is controlled by many factors. The mechanical weakness of a fissured material is clearly of importance but the pore pressure changes and local alterations in water content also play a significant role.

The authors cannot agree with Mr. K. A. Fern (p. 440). Their impression is the reverse, namely that engineers and geologists are inclined to think of grouting for the solution of groundwater problems, and very few of them think of drainage measures. This is very true in dams, where grout curtains have often been prescribed in cases where drainage relief wells would have been more effective and economical in controlling pore-pressures (or 'uplift pressures'). It must be remembered that in many groundwater problems it is not the volume of flow which is significant, but the water pressure at various points and the exit gradient.

There are cases where grouting may be useful in slope stability problems. However, Mr. Fern may be inviting danger if he uses grouting in an attempt to stabilize slopes in which high seepage pressures are a significant factor. He may well reduce the flow of water, but at the same time allow water pressures to build up behind his grouted zone, hence reducing effective stresses and precipitating failure.

Mr. S. C. Brealey: I agree with Mr. R. Glossop (p. 434) that sludge from reverse-circulation bore-holes should not be the only source of formation samples if it is possible to obtain undisturbed samples. However, these samples give a good indication of the aquifer material, except that any cementing matter is broken down. The physical properties of the formation, however, are not fully indicated. Comparative deductions can be made, for instance for the selection of gravel packs and screens. Undisturbed samples, upon which measurements can be made, may be difficult if not impossible to obtain, for instance at depths up to 400 ft in the case of Neyveli, and the expense and delay may not be justified for the limited purpose under discussion. Furthermore, laboratory testing could never be relied upon by itself and only field pumping tests can give reliable information of the large-scale behaviour of an aquifer.

In response to Mr. Glossop's question as to whether any difficulty had arisen due to the precipitation of iron salts in the filters in the wells, the answer is that it was not experienced at Neyveli. In any event, the coarse gravel packs and large slot openings would have made any such encrustation less important than would have been the case with wire-wound screens or fine-grained filters.

I am glad that Professor R. A. L. Black took issue (pp. 435–6) in regard to a rather careless remark about the stability of rock slopes. The problems of dealing with unconsolidated material are always difficult where water is present, but only in certain circumstances with hard rocks. Nevertheless, I agree with Professor Black that the analysis should be made in all cases. I am perhaps fortunate in not having encountered such a problem personally as yet.

In reply to Mr. H. B. S. Dales (pp. 437–8), in the case of Neyveli the aquifer was so unconsolidated, and at a depth of 200–400 ft, that coring was not attempted. In any case it appeared doubtful that the information obtainable from cores could have been of much value compared with pumping test data, i.e. actual behaviour under flow conditions.

With regard to the minimum diameter of pumping-test bore-holes in the case of loosely consolidated sands, the reverse-circulation method of drilling enables a large-diameter hole to be drilled almost as easily as a small-diameter hole and gives better hydraulic conditions, which may be limiting. The diameter of the casing is determined by the diameter of the submersible pump to be used. Using a 12-in diameter pump the casing would be at least 15-in diameter and the bore-hole, say, 22 in. Centralizers must be used to ensure a uniform gravel pack.

In well consolidated permeable beds the casing need only extend to the pump location and the bore-hole below that depth might be left uncased. The diameter of the bore would then be only a few inches greater than the outside diameter of the casing as there would be no necessity for a gravel pack.

The assessment of the groundwater problem in any virgin area is a nice exercise in economics and probabilities. A preliminary pumping test using, say, an airlift pump and at least one observation hole would give enough information if there were no large-scale anomalies or boundaries. Fissured ground should behave generally like porous ground provided that the scale is large enough.* Stuart describes the results of test pumping and dewatering in an underground mine in

---

*STUART, W. T. Pumping test evaluates water problem at Eureka. *Min. Engng*, (*Trans. Amer. Inst. Min. Engrs*, **202**), Feb. 1955, 148–56.

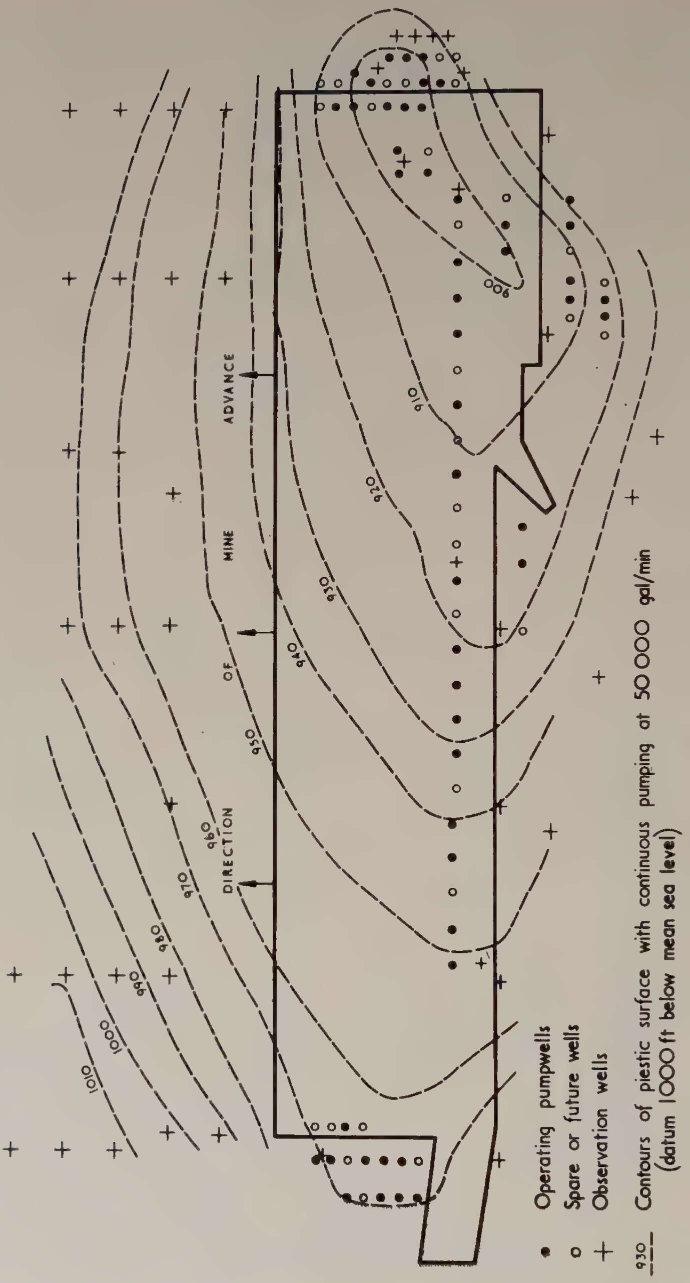

DIRECTION OF MINE ADVANCE
900
910
920
930
940
950
960
970
980
990
1000
1010
● Operating pumpwells
○ Spare or future wells
+ Observation wells
930 Contours of piestic surface with continuous pumping at 50 000 gal/min (datum 1000 ft below mean sea level)

Fig. B. Development pumping at Neyveli (October 1964).

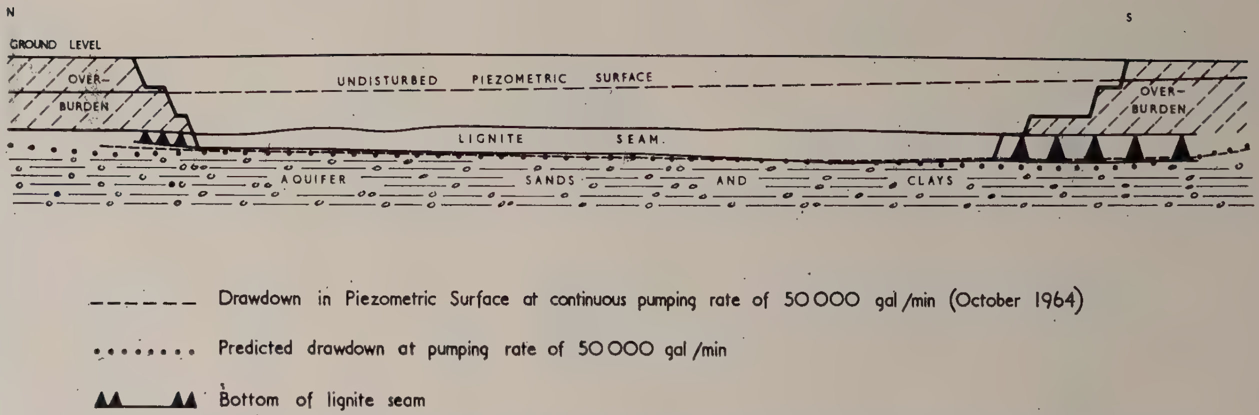

N
S
GROUND LEVEL
OVER-BURDEN
UNDISTURBED PIEZOMETRIC SURFACE
OVER-BURDEN
LIGNITE SEAM.
AQUIFER SANDS AND CLAYS
Drawdown in Piezometric Surface at continuous pumping rate of 50 000 gal/min (October 1964)
Predicted drawdown at pumping rate of 50 000 gal/min
Bottom of lignite seam

Fig. C. Longitudinal section of Neyveli mine showing drawdown achieved by development pumping (October 1964).

faulted limestones, shales, etc. It is safer to achieve the full drawdown required over at least a small area (as at Neyveli). Misleading results are due to smallness of scale in pumping rate, drawdown, areal extent and time, but exactly how far to go is a matter of judgement.

To reply to Mr. R. O. Morris and Mr. K. A. Fern (pp. 439 and 440), freezing and various grouting methods were considered in the case of Neyveli and rejected on account of the astronomical values involved. The volume of the box-cut was approximately 27 000 000 yd^3. The aquifers which would have had to be sealed lay at a depth of over 200 ft and were, in fact, hundreds of feet thick (the full depth was never ascertained as it was of no immediate interest, but sands continued to over 900 ft). Also, the beds were thought to be lenticular, i.e. without continuous aquicludes which could have served as floor seals. The problem of sealing off these beds, both laterally and vertically, was therefore regarded as impracticable. Furthermore, the 'longwall' advance of the face would have meant either re-establishing the seals at intervals or sealing the whole working area—an even greater problem. In the case of shaft-sinking, tunnelling and the smaller excavations used in civil engineering works, these methods have, of course, considerable application.

Figs. B and C show a later situation at Neyveli (October 1964) than Figs. 13 and 14 (May 1963) in the paper. Fig. B shows more pumps operating *inside* the excavation as locations become available so as to obtain the required drawdown with the minimum pumping rate. Fig. C compares the drawdown actually achieved with that predicted from Fig. 11.

GROUP IV—OPENCAST MINING
(bedded deposits and general questions)

Session 6—Planning and Control (Papers 20–23)

Chairman: Professor J. C. Mitcheson *Rapporteur:* Mr. R. N. Pryor

622.341.1 (661.2)
622.271.3:622.341.1 (661.2)

Development of the Kedia d'Idjil Orebodies

(S.A. des Mines de Fer de Mauritanie) MIFERMA
Islamic Republic of Mauritania

J. AUDIBERT, M.I.M.M.
Ingénieur en Chef au Corps des Mines
General Manager of Miferma

P. CARUEL
M. Société des Ingénieurs Civils de France, Ingénieur des Arts et Métiers
Engineer in charge of planning, Miferma

A. CHOUBERSKY, M.I.M.M.
Consultant to Miferma

SYNOPSIS

Planning and carrying out of the Kedia d'Idjil enterprise of S.A. des Mines der Fer de Mauritanie, from 1951 until the first ore boat was loaded in June 1963, are described in this paper.

The mineralized area, principally high-grade hematite ore, of the Kedia d'Idjil has been classified into eight deposits. Two, Tazadit and F'Derik, have been explored in detail; another, Rouessa, has been partially explored. The environment, ore types and estimated reserves are briefly described. The history of the development and early exploration is outlined and the difficulties of terrain, water supply, labour, resources and wear on equipment are emphasized.

Details of the project having been submitted to the World Bank in 1959 and accepted, Tazadit was chosen for mining first, then F'Derik, and Rouessa later. Descriptions are given of pit design, equipment, transport of ore to rail, ore grade, mine target, services, water and power requirements, new town, and the surveying and planning for the construction of a 650-km railway to Port Etienne.

The Kedia d'Idjil rises in a succession of black cliffs to an altitude of 900 m at latitude 22° 41′ N and longitude 12° 13′ W (Fig. 1).

Between the range and the coast extends a dun-coloured rocky plain which drops from an elevation of 410 m to a 20-m cliff along the Atlantic. It is dotted with isolated peaks. Dry water courses, picked out by sparse scrub, wander from the Kedia to a vast salt pan situated 40 km to the north.

The inhospitable coast is cut by two remarkable anchorages, Villa

Cisneros in Spanish Rio de Oro and the Baie du Lévrier in Mauritania (Fig. 2).

The climate is Saharan. For eight months of the year dry northeast winds blow in gusts of up to 100 km an hour and keep the temperature below 30°C. Occasional rains in January mark the northern winter. In summer the temperature exceeds 50°C and in August the tropical rains bring occasional cloudbursts. The 'Irifi' (Harmattan) blows at all seasons with accompanying dust storms and oppressive conditions. The daily temperature range the year round is 20°C. Average rainfall is 6 cm a year, but no appreciable rain has fallen around the range since 1953.

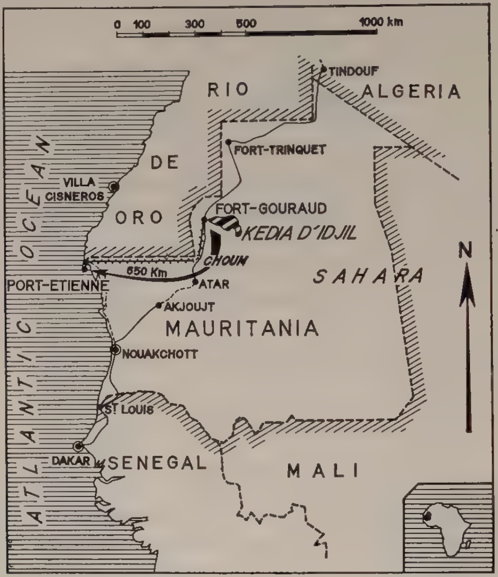


Fig. 1.—Situation map.

Mauritania has an area of 670 000 km^2 and a population of 900 000; 300 000 are settled along the banks of the Senegal River and the remainder roam the country with their herds of camels and goats in search of the temporary pastures occasioned by local rainfalls.

Building is still going on in the capital, Nouakchott, and a small fishing industry exists at Port Etienne on the Baie du Lévrier. All other settlements lie along the old caravan trails—wherever a supply of water appeared to ensure stability to life.

A dirt road connecting Casablanca to Dakar passes at the foot of the Kedia and a network of dirt tracks has been built by the French Army.

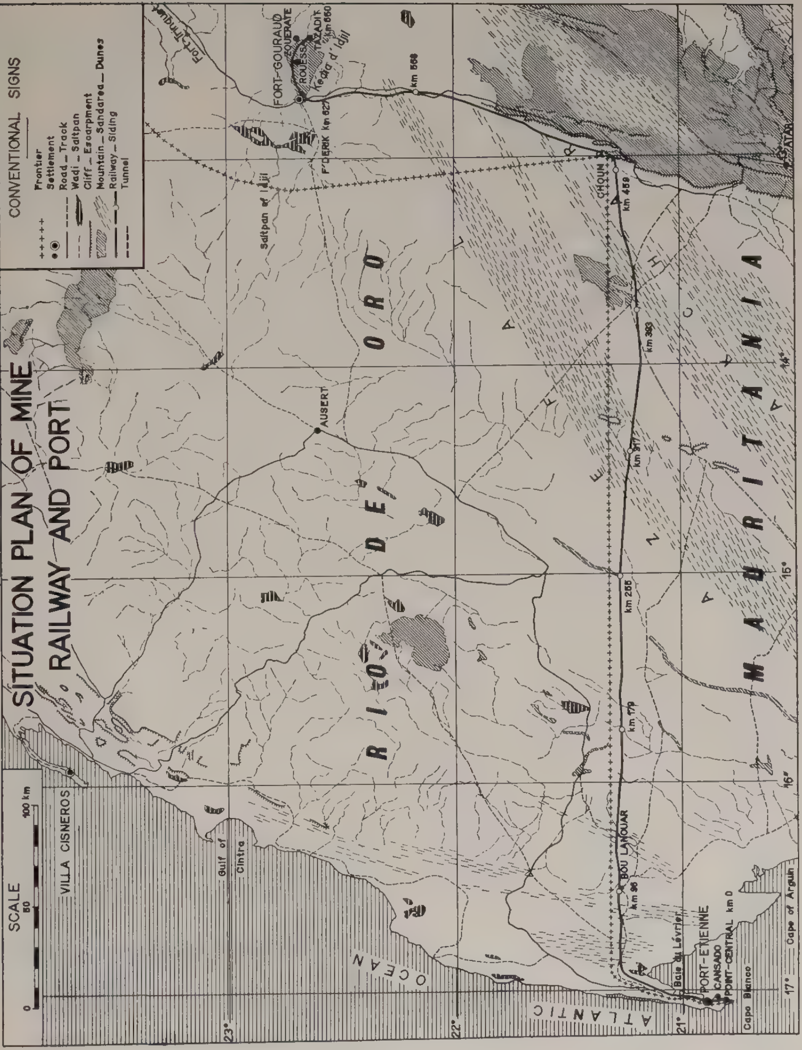

SITUATION PLAN OF MINE
RAILWAY AND PORT
CONVENTIONAL SIGNS
Frontier
Settlement
Road _ Track
Wadi _ Saltpan
Cliff _ Escarpment
Mountain _ Sandared _ Dunes
Railway _ Siding
Tunnel
SCALE
0
50
100 km
VILLA CISNEROS
Gulf of Cintra
ATLANTIC OCEAN
RIO DE ORO
MAURITANIA
AUSERT
Saltpan of Idjil
F'DERIK km 627
FORT-GOURAUD
ZOUERATE
ROUESSA
TAZADIT km 650
Kedia d'Idjil
Fort Trinquet
km 558
CHOUM
km 459
ATAR
km 393
km 317
km 255
km 179
BOU LANOUAR
km 90
Baie du Lévrier
PORT-ETIENNE
CANSADO
POINT-CENTRAL km 0
Capo Blanco
Cape of Arguin
23°
22°
21°
17°
16°
15°
14°

Fig. 2

None of these can carry sustained heavy traffic, and Port Etienne has no road connecting it with the hinterland.

The nearest industrial centres are Dakar and Agadir and the bulk of the population has had little contact with modern life.

GEOLOGY

The plain consists of early Precambrian granites which disappear westwards below recent sediments and eastwards below Cambro–Ordovician cliffs.

The roots of a late Precambrian formation wind through the plain over an area of 100 000 km^2. They consist of white quartzites, shales and banded hematite and magnetite quartzites (BHQ). Subvertical and contorted, these strata stand out as inselbergs wherever they have been able to withstand erosion.

The Kedia d'Idjil is the largest of these inselbergs (Fig. 3). It is shaped as a right-angled triangle with its apex pointing NE, a 25-km side facing N and a 10-km side facing E. Its contours are very rugged.

The northern and eastern flanks of the range consist of beds of shale and BHQ compressed into tight synclinal folds which cover an area up to 2 km wide. The heart and the southern flank consist of a great thickness of peculiar breccia.

The formations have been disturbed by widespread dragfolding and blockfaulting and a series of N–S shearing zones has cut the range into compartments separated by deep cañons.

Within the BHQ fifty outcrops, covering a mineralized area of 900 000 m^2, have been classified into eight deposits. Two of these, Tazadit and F'Derik, have been explored in detail. A third, Rouessa, has been partly explored.

The ore is predominantly a hematite containing over 58 per cent Fe. The BHQ averages 45 per cent Fe. Materials between these two values are scarce.

The ore is of three types: (*a*) a very hard, compact material of homogeneous structure with a density *in situ* of 4·5; (*b*) a soft 'plaquette' material with a density *in situ* of 3·1 which consists of hematite lamellae more or less embedded in powdery quartz; and (*c*) a hard, banded material with a density *in situ* of 3·5, probably derived from (*b*) but more porous and friable than (*a*).

The BHQ has a density *in situ* of 2·7 to 3·0 according to its consistency, which can vary from a tough and abrasive material to soft plaquettes merging into type (*b*) ore.

The F'Derik outcrops cover an area of 74 000 m^2 which extends 900 m along the crest of the westernmost spur of the range above the

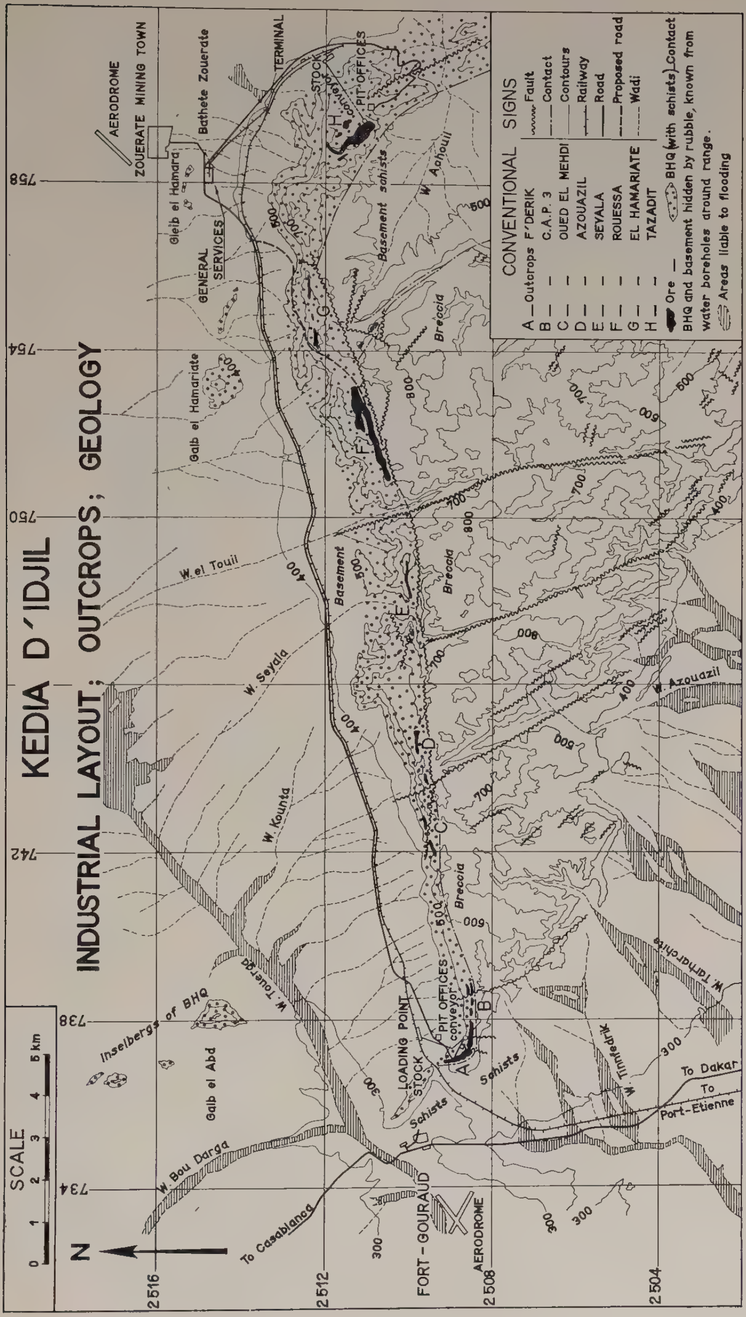

KEDIA D'IDJIL
INDUSTRIAL LAYOUT; OUTCROPS; GEOLOGY
SCALE
0 1 2 3 4 5 km
N
734
738
742
750
754
758
2516
2512
2508
2504
AERODROME
ZOUERATE MINING TOWN
Bathete Zouerate
TERMINAL
STOCK
conveyor
PIT OFFICES
Basement schists
W. Aahouil
Gleib el Hamara
GENERAL SERVICES
Galb el Hamariate
W. el Touil
Basement
Breccia
W. Seyala
W. Kounta
W. Touerga
Inselbergs of BHQ
Galb el Abd
W. Bou Darga
To Casablanca
FORT-GOURAUD
AERODROME
LOADING POINT
STOCK
Schists
W. Tinfedrik
W. Tarharchite
W. Azouazil
To Dakar
To Port-Etienne
A
B
C
D
E
F
G
H
300
400
500
600
700
800
CONVENTIONAL SIGNS
A — Outcrops F'DERIK
B — " C.A.P. 3
C — " OUED EL MEHDI
D — " AZOUAZIL
E — " SEYALA
F — " ROUESSA
G — " EL HAMARIATE
H — " TAZADIT
Ore —
BHQ (with schists) Contact
BHQ and basement hidden by rubble, known from water boreholes around range.
Areas liable to flooding
Fault
Contact
Contours
Railway
Road
Proposed road
Wadi

Fig. 3

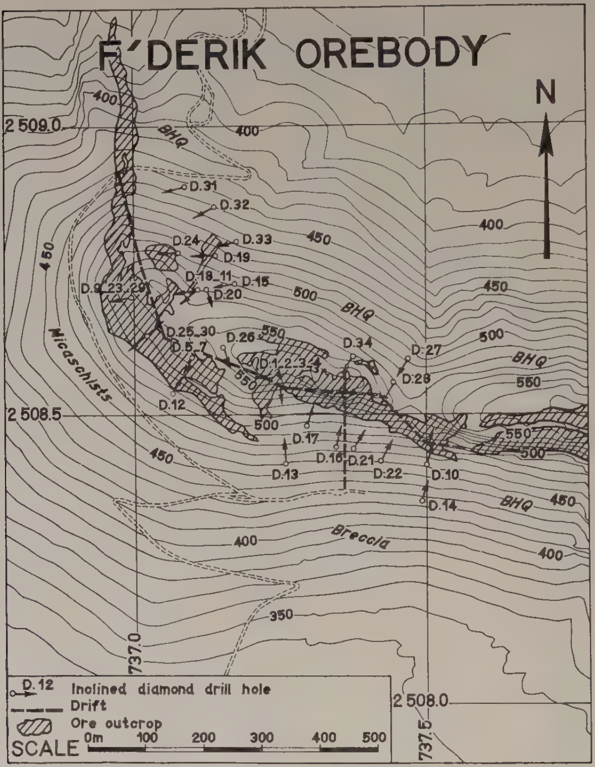


Fig. 4.—Plan of F'Derik orebody.

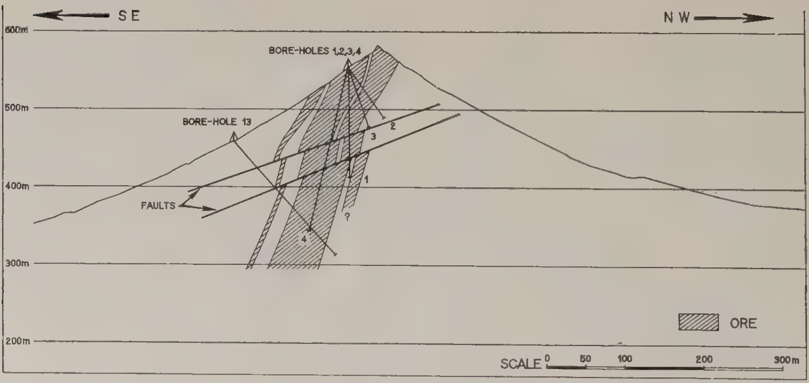


Fig. 5.—F'Derik: vertical section.

500-m contour (Figs. 4 and 5). This spur curves northwards and is of triangular section with slopes of 30–40°. Its summit is at 585 m.

The ore, entirely of type (*a*), is distributed in a series of irregular overlapping lenses which tend to finger out into BHQ. It has been followed down to a depth of 250 m below mean outcrop level, where its thickness still appears to be up to 40 m.

The orebody is disturbed by a series of crossfaults and subhorizontal faults. A barren shearing zone 50 m wide cuts it into two wings and throws the eastern wing 100 m to the north. The eastern wing dips 70°S and the northern wing 60°NE, conformably with the surrounding rocks.

The Tazadit outcrops (Figs. 6 and 7) cover an area of 206 000 m^2 along an inner spur of the range at an altitude of 700 m. The main outcrop trends 1100 m to the NW and then continues as narrow stringers.

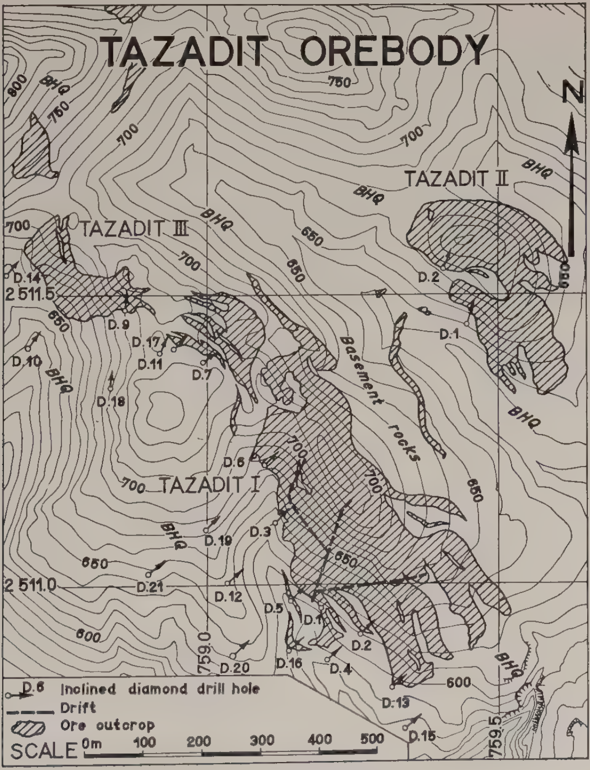


Fig. 6.—Plan of Tazadit orebody.

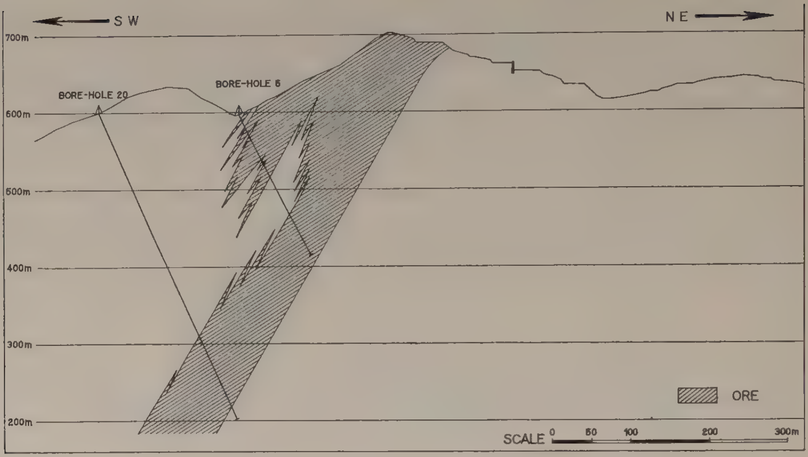


Fig. 7.—Tazadit: vertical section.

To the south it passes rapidly into BHQ. The ore dips 60° SW, conformably with the surrounding rocks.

The hanging-wall has been scoured off the southern end of the outcrop to an altitude of 590 m and has exposed a 200-m width of ore. At its northern end the outcrop narrows to 100 m, distributed between several lenses. The hanging-wall has not been scoured away and overlies the bulk of the ore. The hanging-wall consists of tough abrasive BHQ, and the footwall contains alternate beds of soft shale and BHQ.

Along the footwall extends a lens of type (*c*) ore, 700 m long and over 100 m thick. Two bore-holes have followed it down 400 m below mean outcrop level where it still retains its thickness. It is continued to the north by two smaller similar lenses offset towards the hanging-wall.

Between the lenses of hard ore and the hanging-wall lie large irregular pockets of type (*b*) ore which merge into BHQ some 200 m below mean outcrop level. Their grade is variable with soft ore passing into soft BHQ and inclusions of hard materials. Hard BHQ penetrates into the heart of the orebody along the strike, but without disturbing the pattern of mineralization, and there is no faulting.

A second outcrop, 250 m long by 125 m wide, covers a small hill 350 m from the footwall of the main orebody. Its characteristics are very similar to those described above, but the ore appears to peter out 150 m below outcrop level.

Reserves are now estimated as follows:

(in thousands of metric tons)

	Proved	*Probable*	*Possible*	*Total*
F'Derik	19 830	20 000	?	39 830
Tazadit	86 930	30 000	20 000	136 930
Rouessa	14 500	15 000	50 000	79 500
Other orebodies	—	—	35 000	35 000
	121 260	65 000	105 000	291 260

These figures do not include known beneficiating material and the estimate for 'other orebodies' is probably conservative.

HISTORY OF THE DEVELOPMENT AND EARLY EXPLORATION

High-grade ore was reported from the area early in the century, but was only of academic interest until the Bethlehem Steel Corporation examined the outcrops in 1948. The property then remained dormant until an Anglo–Franco–Canadian mission recommended exploration in 1951 and 'Miferma' was founded as a prospecting company early in 1952 with a capital of 30 000 000 CFA (U.S. $120 000). Work started immediately to determine the probability of 100 000 000 metric tons of ore containing over 60 per cent Fe and of an adequate supply of water which could justify further expenditure. The ore was to be railed 411 km through Spanish Rio de Oro and shipped via Villa Cisneros.

By 1954 1250 m of drilling, 460 m of tunnelling and some surface trenching had been completed on F'Derik and Tazadit; a 32-m water well, carrying a drainage gallery, had been sunk near F'Derik and a 120-m bore-hole sunk in the floor of the cañon at the foot of Tazadit. All this work gave positive results.

A temporary camp with prefabricated houses and aluminium huts had been put up, equipped with mess, club house, electricity and water. A small workshop had been installed and the main orebodies connected by car tracks.

In 1954 it was decided to switch work to a third orebody, Rouessa, where mapping had shown an area of 377 000 m^2 of ore. Drilling continued on this orebody until mid-1955 when it was switched back to Tazadit and F'Derik as it had shown a preponderance of soft ore, which was then at a discount.

Few precedents existed at that time for large-scale exploration under similar conditions and much had to be learnt through experience.

The major difficulties were:

Lack of water, which was limited to 10 m^3 of sweet water and 10 m^3 of brackish water per day for the requirements of two drills, two tunnelling teams, 30 Europeans and 300 Africans between workmen and village, until the middle of 1953 when the company well gave an extra 30 m^3 a day. Shortage of water hampered all but the last two years of development.

Difficulties of supply: Local resources were limited to goat and camel meat. Everything was brought in by road or plane from Dakar and Agadir, a distance of some 1000 km. Transport costs were around 35 000 CFA (U.S. $140) per ton, losses were considerable and arrivals slow and erratic.

Abnormal wear on equipment, principally due to the fine abrasive dust which penetrated equipment considered as airtight.

Lack of local labour, which also suffered from malnutrition. It required many months of regular rations and discipline before it could be usefully employed. Meanwhile labour was imported from Senegal.

Considerable turnover on expatriate labour: The harsh and isolated living conditions led to many cases of moral breakdown and men had to be flown home.

Route and port

During 1952–53 an independent firm was entrusted with the survey of the railway route through Rio de Oro and of Villa Cisneros bay. Meanwhile the French Government, anxious to see the ore shipped through a Mauritanian port, were investigating the possibilities of Port Etienne.

It was found there that a 20-m deep channel penetrated from the open sea far into the bay 400 m off the coast of the peninsula and was scoured by the tidal currents, whereas at Villa Cisneros the peninsula was inaccessible by rail and a channel had to be cut across the bay through a series of shallows, two of less than 5 m deep, and could not come closer than 1000 m from the coast. To accommodate a modern ore carrier this channel would require several million cubic metres of initial dredging and, as it ran across the tidal currents, heavy maintenance dredging to keep it open.

As against this, the railway to Villa Cisneros was 230 km shorter than the all-Mauritanian route and did not cross any obstacles, whereas the railway to Port Etienne had to cross three areas of sand dunes and, if it were to stay within Mauritania, a 250-m cliff.

The relative merits of the two routes were compared during 1955–56 and the advantages of Port Etienne as a harbour were found to outweigh the disadvantages of a longer railway route.

Water

Work still continued on the Kedia, but the main effort now turned to exploration for water.

The Kedia contains several catchment areas, some covering up to 30 km^2. Each has a single outlet on to the plain through a cañon cut along a shearing zone. The 6 cm of rainfall a year on such a catchment area represents 1 800 000 m^3 of water. Admitting a 95 per cent loss through runoff and evaporation, there still remain 90 000 m^3 it can be hoped to tap.

Bore-holes in the alluvial fans at the mouth of the cañons proved the existence of an underflow, but the yields were insignificant owing to the clayey nature of the soil. Bore-holes in the floors of the cañons, similar to that sunk in 1953 but 12 in. in diameter, found water 50 m below the collars and yielded several cubic metres an hour, adequate for immediate needs. (Subsequently, these holes were deepened to 120 m and the grid extended to three catchment areas. Present yields are around 15 000 m^3 a month and the level appears to have become stabilized some 70 m below the hole collars. It is realized that these reserves correspond to limited and probably disconnected areas and are uncertain. They are moderately drawn upon and kept as a 'strategic' reserve.)

Simultaneously, the French Government carried out drilling and geophysical exploration around an old well, 100 km inland from Port Etienne, situated in sedimentary formations similar to those containing the Senegal water-table. This brought to light reserves which are now estimated at 1 000 000 000 m^3.

Report and estimates

By 1957 4200 m of drilling and 300 m of tunnelling had been completed on F'Derik, 1500 m of drilling and 600 m of tunnelling on Tazadit, and 1000 m of drilling on Rouessa. Port Etienne had been chosen as the shipping port and information was sufficient to prepare an overall report which could be submitted to the International Bank for Reconstruction and Development in view of a loan. Expenditure up to this date was 412 500 000 CFA (U.S. $1 650 000). The report was favourably received and the Company encouraged to increase its capital to 1 237 500 000 CFA (U.S. $4 950 000) in order to carry out the detailed investigations necessary to prepare the construction programme, a long-term programme of operation and the corresponding financial estimates.

It was also realized that, in order to compete with other mines then being developed, the commercial grade of ore would have to be raised to around 64 per cent Fe, 3 per cent SiO_2, and 1 per cent Al_2O_3, and that other constituents would have to be closely controlled.

Detailed project

The staff in the field and at head office was expanded and the planning completed by the end of 1959 when a detailed project was submitted to the World Bank, comprising:

estimates of reserves;
a design of the pits and of their approaches;
proposals for the evolution of the pits;
specifications for all plant and equipment on mine, railway and port;
a detailed plan and timing of construction;
a calendar of investments, overheads and interest;
an estimate of yearly outputs and costs covering 18 years;
estimated operating accounts covering 7 years;
a treasury forecast;
regulations and basic prices to be applied by contractors; and
details of services to be rendered by the company to contractors.

The International Bank granted a U.S. $66 000 000 loan on 17th March 1960, the French Government lent 155 000 000 New Francs (U.S. $31 000 000) and the capital of the Company was increased to 13 300 000 000 CFA (U.S. $53 300 000).

Work started immediately and the first ore boat was loaded in June 1963.

THE MINE PROJECT—TAZADIT

At the mine, work was handicapped by the rugged terrain and difficulties of supply.

By 1959 results were sufficient to prepare the World Bank report, but still inadequate for proper planning. The World Bank accepted this situation because (*a*) adequate margins had been taken in estimating; (*b*) the possible errors were small when compared with the investments and costs on the port and railway; and (*c*) allowances had been made for the problems which remained unsolved.

Tazadit was chosen for mining to start because it contained the largest proved reserves and because the pit area was large enough to carry an output of over 4 000 000 tons a year. F'Derik will be equipped for 1 000 000 tons a year now that the railway has cut overheads, and Rouessa will be developed once F'Derik is in production.

Pit design

Exploration had outlined the Tazadit orebody down to the 440-m level, shown up the structural details, localized the areas of high- and low-grade material, and proved tonnages and grades sufficient to justify investment. The pit (Fig. 8, Plate I) was designed as a V-shaped cut to go down to this elevation.

There was ample evidence that minable ore continued below the proposed bottom of the pit, but information was insufficient to extend it downwards. This eventuality was accepted because it will have little influence on the conduct of the first years of operation and because the ore from these levels was not included in the reserves declared to justify the operation. Provision was made to take the pit down if required.

The pit shape is governed by the 60° dip of the orebody and by its strike along a hog-backed ridge, flanked on both sides by deep ravines with several million tons of ore exposed over a height of 130 m along its southern end.

The slope of the pit walls was chosen empirically by comparison with other mines. The hanging-wall, in massive BHQ dipping outwards, will be benched back 13 m every 20 m, while the footwall, underlain by soft schists, will be benched back 10 m every 10 m. Provision has been made for service roads along the pit sides.

A 'bun' of BHQ on the hanging-wall partly covers the northern end of the orebody but, once this has been levelled off, stripping will be reduced to benching back the walls of the pit, and the average ratio of 2 of waste to 1 of ore will be adhered to as closely as possible throughout the life of the mine.

The disposal of waste down to the 440-m level raises no problems. Above 580 m the pit has direct access to a large valley which lies beyond the hanging-wall; down to the 500-m level this valley can be reached by trenching, and the waste from the 440-m level can be hauled with a 60-m lift. Several waste dumps have been laid out at different levels to reduce haulage as much as possible.

Plans below this level have not been finalized. The choice will lie between a skip and a haulage road according to the volume of materials to be moved.

Equipment

The size of equipment was determined by comparison with other mines and the choice guided by the principles of using the largest equipment compatible with the selectivity required, in order to reduce manpower; and ensuring that equipment was powerful and strong enough to handle heavy and tough material in difficult conditions.

For stripping, three Ward-Leonard 8-yd^3 shovels were chosen as being the biggest in standard use in many parts of the world. For ore, the choice lay between 6-yd^3 and $4\frac{1}{2}$-yd^3 shovels, possibly equipped with smaller buckets. Two Ward-Leonard 6-yd^3 shovels were chosen. Shovel bodies are pressurized and the cabs air-conditioned. Three $2\frac{1}{2}$-yd^3 diesel shovels left over from development are used on narrow faces. To allay dust along the transport line $1\frac{1}{2}$ per cent of water is sprayed ahead of the shovel buckets when digging.

Track and tyre-mounted bulldozers clean the benches and dumps.

The trucks chosen were the largest possible in standard production. Ore and waste are hauled in twenty-two 60-ton trucks equipped with oleopneumatic suspension and hydraulic retarders. They are supplemented by sixteen 27-ton trucks left over from development.

Haulage roads are being widened to 15 m and are kept to a maximum 8 per cent grade. They are regularly sprayed with lignosulphonate.

Jet piercing is uneconomic under local conditions and ore and rock are drilled by percussion drills. The drills are of the 'down-the-hole' type and can drill a 43-ft hole at any angle from vertical to horizontal. A crawler-mounted chassis carries a column 33 ft 7¾ in. long, adjustable at any angle.

The rotation drive is set on a cradle which moves up and down the column under the action of a hydraulic feed. It transmits the rotary movement and a constant pressure, which can be adjusted, through the string of rods to the hammer which is situated immediately behind the bit.

The rods are of 5 in. outer diameter, round and hollow, in 20-ft lengths threaded at both ends (male and female) and are rapidly put into position by means of a pivoted rod-changer and hoist. Coupling and uncoupling is done by reversing the movement of the rotation drive.

The hammer is cylindrical, 4½ ft long by 5½ in. outer diameter. The bore is 3¼ in. and the stroke 5 in.

The tungsten carbide crossbits are fitted directly into the hammer bit-holder. Their diameter is 6 in.

Bit efficiency varies from 80 m in hard ore to 10 m in very hard BHQ. The bits can be reground 35 times, thus giving a total bit life of between 2800 m and 350 m.

The drills are equipped with dust collectors and an adjustable water feed through the bit. All cuttings and dust can thus be collected for sampling without undue loss, the silicosis hazard is reduced, and by regulating the flow of water the sides of the hole can be moistened sufficiently to prevent collapse in dry unconsolidated ground.

Power for all the above requirements is supplied by a 600-ft^3/m diesel-driven compressor mounted on rubber tyres. During movements the drill hauls its compressor. Independent compressors were chosen in preference to built-in ones so as to economize on the quantity of 'stand-by' equipment required.

These units drill, including movement, in an 8-hour shift: in hard ore, seven 12-m holes; in average BHQ, four; and, in very hard BHQ, two.

Blasting was initially carried out by means of classical explosives. Then ammonium nitrate of various grades and grain sizes was tried out, mixed with varying proportions of fuel oil in a concrete mixer housed near the pit. Trials are now being run with commercial 'slurries' containing TNT or NG, placed in the bottom of the hole in varying quantities, the rest of the column consisting of fine-grained ammonium nitrate mixed with 5–6 per cent fuel oil.

The blasting pattern is also being adjusted so as to obtain a fragmentation which will give maximum efficiency to the operation of the 6-yd^3 and 8-yd^3 shovels, and which, for the ore, will not 'ride' on the 54–74-in gyratory crusher, while avoiding an undue production of uncontrolled fines.

A 5-m by 5-m spacing appears to be satisfactory for the ore, while 5·5-m by 5·5-m is being tried out in BHQ.

Consumption of explosive is 100 g/ton in ore and 180 g/ton in BHQ.

Bench height is kept to 10 m for purposes of grade control and the pit will go down two benches a year. Mining will progress along the strike in order to obtain better separation between ore and waste.

The ore is hauled to a 54–74-in gyratory crusher, which is set at 6 in., sunk into the footwall at an altitude of 640 m immediately beyond the possible pit limits. From the crusher the ore falls into a 200-ton hopper and is delivered by a variable-speed feeder on to the conveyor system.

To reduce abrasion the sides of the hopper were lined with massive blocks of white quartz, quarried locally. These were not trued, but embedded in concrete and assembled so as to give a smooth surface inside the hopper and reduce as much as possible the area of concrete exposed to the ore.

In order to protect the conveyor line and to avoid overflowing of the hopper, the hopper is equipped with three radioactive controls: one, at the discharge gate, closes the gate when the hopper is empty and prevents blocks falling straight through on to the pan feeder; one, two-thirds of the way up, slows down the speed of the feeder, and the last, at the top, gives warning that the dumping must be suspended.

Pit-to-railway conveyor

The major problem was to find the best means of bringing the ore out of the pit and delivering it to the railway in the plain. Several methods were examined, but

(*a*) the railway could not be brought up to the pit because of the steep approaches;
(*b*) a ropeway could not achieve the desired output and was liable to hazards from the frequent violent winds; and
(*c*) surface conveyors would have to climb the outer ridge of the range and then descend along the outer spurs and valleys.

The most economical solution as regards both investment and operation was to carry the ore through a descending tunnel on a compensated belt. The final route was chosen because it was the shortest, it was situated throughout in good rock, and came out in an area well suited for the railway terminal and possible future extensions.

The conveyor line is 1800 m long and lowers the ore 194 m to the stockpile area at the foot of the range. It comprises three transfer points each equipped with powerful dust extractors. Between the first and second points the inclined conveyor, 1400 m long, runs through a 700-m tunnel cut through the flank of the range.

The main belt, 1·20 m wide, is steel-cord reinforced, and runs at 2 m/sec for a rated output of 1800 t/h; 0·44 kWh are generated per ton carried. This is fed into the grid through a resistor which absorbs overloads. The driving heads act as synchronous motors for starting, then as synchronous generators for braking. Mechanical safety brakes act automatically in case of power failure.

At the second transfer point a take-off has been built in for a future treatment plant.

At the third transfer point, a sampling tower diverts 2 per cent of the load and reduces it to a 10-kg sample. Excess material returns to the conveyor.

The ore is here transferred to the stacker belt which raises it on to the tower. The latter is equipped with two loading arms, each designed to build a triangular pile 16 m high by 40 m at the base and 185 m long, capable of containing 120 000 tons. The tower traverses between the piles. Each loading arm carries an electronic probe which raises it as the pile grows and moves the stacker forward as the pile extends.

Loading

Reclaiming is done by a crawler-mounted wheel reclaimer. It carries eight 660-litre buckets on a 6·6-m diameter wheel at the end of a 23-m adjustable arm, and takes a vertical cut, thus achieving some mixing of ore. The ore then falls on to the distributor arm and through a hopper on to a reclaiming belt which lifts it to the top of a 800-ton loading bin where it is distributed by a shuttle conveyor.

The loading bin straddles the railway track and carries two groups of five loading gates equipped with air-operated guillotine cutoffs. The flow is controlled by four weighbridges, one under each car bogey, but final closure is controlled by hand. Cars, moved forward in sections of fourteen by a shunting locomotive, are loaded in pairs.

The present practicable output of the reclaiming line is 2500 t/h, but it can easily be increased to 4000 t/h.

Conveyor terminal and crushing plant

Several locations were also examined for the pit terminal of the conveyor and the crushing plant. The crusher was finally placed as close as possible to the centre of gravity of the orebody so as to reduce the number of ton-kilometres to be carried before crushing.

Some 10 000 000 tons of ore will be delivered to this site without lifting and a further 10 000 000 tons down to an elevation of 600 m, with only a 40-m truck lift.

After visits to various mines and talks with operators and manufacturers it was decided that the surest and cheapest way of lifting the ore below the 600-m level would be to put in a skip along the footwall at a slope of 45° and to extend it by 40-m increments.

The necessary headframe and hoist have been allowed for in the design of the crusher house.

Several methods of lifting the ore from the 300-m level to the 440-m level are being investigated. The decision will be taken when more information is available concerning these levels.

Grade control

Another series of problems concerned the quality of the ore, selectivity of mining and ore grading.

From the start of the operation, results were studied not only as regards the structure of the orebody and the distribution of its various components, but also as regards the interrelationship of these components and its influence on grades.

All assay results were plotted on frequency distribution charts, according to the chemical element, the origin of the sample and the methods of sampling. Whenever possible, workings were resampled by different methods, results calculated on alternate samples and checks made for possible bias. The frequency curves showed two groups of materials: the ore group, over 58 per cent Fe, and the BHQ group, between 45 per cent and 35 per cent Fe. Intermediate materials corresponded either to interfingering of hard ore and BHQ or to incomplete leaching of the soft ores, and were mostly located around the inclusions of low-grade material within the orebody or along the walls.

SiO_2, Al_2O_3, ignition loss contents were plotted against the Fe content. It was found that these components constituted 99·4 per cent of the ore group materials and that their interrelationship was sufficiently close to be used for forecasting.

The other constituents of the ore are distributed according to a geographical pattern which is as yet unexplained.

It was further found that the proportions of the constituents varied according to the screen fractions, whether these were natural or mechanically produced. The lump is always higher in grade and purer than the corresponding fines.

All exploration data were then redesigned, using contents curves spaced every 3 per cent Fe, and these plans were used as a basis for the World Bank report.

Ahead of the shovels, the ore is now permanently drilled to a 60-m depth on a 50-m square grid which is moved every year along one of its diagonals, thus achieving a 25-m square grid for the ore a year ahead of the working faces.

All blast-holes are surveyed and the cuttings assayed and assay and geological plans kept for the levels as they are worked out, supported by vertical cross-sections spaced every 25 m. This enables the whole operation to be laid out in adequate detail three years ahead of mining.

Before each blast the mining crews are provided with a diagram showing which materials are to go to the crusher, which to the 'beneficiating' dump and which to the waste dump. The output of each shift is automatically sampled on the outhaul belt and the progress of the shovels during the next shift is corrected according to the assay result.

The present objective is to balance the progress of faces in such a way that the average commercial grade foreseen will be achieved to within ±0·5 per cent of Fe, SiO_2, Al_2O_3 on every 150 000 tons of output. This tonnage will then be homogenized on a blending pile at port.

Service buildings and plant, new town and amenities

Near the pit are situated the pit offices, a maintenance workshop, the transformer station and a water tower.

The 'general services' in the plain comprise the head office, a laboratory, the general store, a workshop equipped for the major overhaul of all equipment and a 11 200-kW power-house (Fig. 3).

Power requirements are considerable because of air-conditioning, pressurizing, cold storage, etc. The power-house and certain workshop areas are pressurized and fed with filtered air. Power is distributed by 30-kV overhead lines and stepped down to 550 and 380/220 V.

Storage tanks for 2800 m^3 of fuels have been built by a supplier under contract to Miferma.

All industrial buildings are serviced by railway spurs and by heavy overhead cranes.

Water is pumped 7 km from two catchment areas to a treatment plant and is distributed by gravity. 500 m^3 a day are drawn from local reserves and a further 500 m^3 are brought in by train from Port Etienne.

The town 'Zouerate' comprises the houses for employees of three types—37 'senior staff', 200 'foremen' and 420 'workmen'. Care has been taken to avoid direct insolation, to provide good insulation and to avoid penetration of dust. Most of the houses are air-conditioned and all have running water, electricity and drainage.

Two schools, a training centre for apprentices, a hospital with an operating theatre, a club, a cultural centre, a cinema, two swimming pools, an area reserved for private tradesmen and an area reserved for government buildings have been provided.

The basic building material is hollow concrete tile with exterior aluminium shuttering to provide insulation. The roofs are terraced and protected by a projecting aluminium shade.

A network of tarmac roads and a telephone/wireless grid connect the pits, general services and town.

THE RAILWAY AND PORT PROJECT

Railway (Fig. 2)

The railway line is 650 km long. It follows the boundary of Rio de Oro and drops from an altitude of 410 m at the eastern extremity of the range to 20 m at Port Etienne.

The obstacles crossed—from E to W—are:

the Choum cliff, impossible to avoid without penetrating into Spanish territory;

the Akchar fixed dunes, extending over 20 km of track, from 5 to 20 m high, spaced every 200–300 m;

the Azefal fixed dunes, extending over 50 km of track, from 2 to 5 m high, spaced every 35–50 m; and

the live, crescent dunes, extending over 50 km of track at the head of the Baie du Lévrier. Up to 30 m high, they move south at a rate of 10 to 100 m a year, in inverse proportion to their height.

A tunnel, 1890 m long, reached by approach ramps, was driven through Choum cliff. The fixed dunes were crossed on embankments. A study of the live dunes showed that the small ones can be destroyed easily whereas, when the few large ones in the vicinity come to threaten the track, the line will be diverted. When cuts were unavoidable they were made wide and shallow and as far as possible directed so as to obtain a scouring action from the dominant winds.

The subgrade is good and the ground flat. Including ballast, but excluding the tunnel, all earthmoving amounted to 5·5 m^3 per linear metre. Good granite and limestone ballast were found locally.

In view of the indeterminate drainage pattern no culverts were put in except along foothills. They will be put in where washouts occur.

The line, which is single track with seven crossing points between terminals, is laid with 0·5 per cent grades against full trains and 1 per cent against empties. There are no curves under 500 m radius. The 54-kg UIC rail is continuously welded between crossing points and is laid on wooden ties over the first 60 km from the coast to avoid corrosion and then on all-steel ties spaced every 60 cm. The permissible axle load is 25 metric tons. Speeds are 40 km/h for the full trains and 60 km/h for the empties.

The rolling stock consists of:

15 line locomotives: C-C, 126-t, 2800 CV, two diesel-electric drives, drawbar pull 38 000 kg, maximum speed 70 km/h;
8 shunting locomotives;
600 ore cars: 77·5-t load, 22·5-t tare, two cast-steel bogies, monobloc cast-steel wheels with hardened rims, sealed roller bearings, automatic couplings;
34 tank cars for fuel and water;
42 flat and sundry cars; and
12 ballast cars.

The diesels draw their air through a series of heavy impregnated filters piled along the sides and roof of the airtight locomotive bodies. The front wheels are equipped with retractable sand chasers.

At full operation two trains a day will run both ways. They will consist of 3 locomotives and 130 ore cars plus tank cars and flat cars (two of them to be reserved for the government) and will carry 10 000 tons of ore.

The caboose at the end of the train is air-conditioned and equipped as a rest car for the relief crew which takes over half way.

Dispatching is done by wireless from Port Etienne and there is a VHF link between locomotive and caboose. Turn-around time is 48 hours.

On arrival at the mine the trains are broken up, serviced, loaded and dispatched in 6 hours. At port the ore cars are split into four sections on parallel tracks. They are uncoupled and moved forward by a shunting locomotive on to two line retarders from which they run by gravity on to a 'barney mule' which lifts them into a rotary dumper with a swing of 170°.

Tipping rate is 54 cars/h. The full cars push out the empties which run by gravity into a checking and assembly yard.

A large workshop carries out railway maintenance and caters for town and port; it is also equipped to carry out specific jobs for the mine. Its capacity is 550 000 man-hours/year. Locomotives are serviced in a three-tier station.

Port (Fig. 9, Plate I)

The port is situated within the Baie du Lévrier, 11 km south of the town of Port Etienne.

Transfer of ore

From the railway dumper the ore falls through a hopper and pan feeder on to a transfer belt which lifts it to ground level and carries it to the stacker belt. The stock lies parallel to the coast and the stacker traverses along the centre line, layering the ore. The stacker has a single rotating arm, 45 m long, which can feed two triangular piles 23 m high by 58 m at the base and 700 m long, each with a theoretical capacity of 1 000 000 tons of ore.

Belts are 1·60 m wide, run at 3 m/sec and deliver at 4000 t/h.

A crawler-mounted wheel reclaimer carrying eight 600-litre buckets on a 6·5 m wheel with a practical capacity of 2500 t/h and an 8-yd^3 long-reach electric shovel with a practical capacity of 1300 t/h are used for reclaiming. The ore is taken in vertical cuts in order to blend it.

Two mobile hoppers deliver the ore on to two outhaul belts running parallel to each pile feeding a collector belt running perpendicularly to the coast.

The ore then passes through a sampling tower and into a 100-t regulating hopper which feeds the pier conveyor, wharf conveyor, and loading tower. Ship loading is thus rendered independent of reclaiming.

The loading tower traverses 166 m. It carries an arm which can be adjusted 16 m horizontally and 10·5 m vertically to load boats between 6 000 and 100 000 tdw (tons dead weight).

For the time being only one wing of the stock and one loading line have been equipped, with a practical loading rate of 2500 t/h.

A *screening plant*, with a feed capacity of 600 t/h, has been put in immediately north of one wing of the stock and equipped with a separate reclaiming line. It will grade the ore at 50 mm, 20 mm and 6·3 mm. Provision has been made to extend this plant and to put in a beneficiating plant at a later date.

The *loading wharf* lies parallel to the coast along the edge of the 20-m channel at the end of a prestressed concrete pier 400 m long and 10 m wide. The wharf is 245 m long by 19·5 m wide and is extended southwards by a 128-m catwalk carried on three dolphins. The catwalk is equipped for pumping and can be used either as an unloading berth for oil tankers or a waiting berth for ore carriers. No general goods will be handled at the ore port.

The wharf is built in two sections. The front consists of a deep concrete beam carrying the mooring bollards and anchored on to eight sheet-pile rock-filled caissons, 13 m in diameter, projecting beyond the quay face and sunk through the sand and silt into the sandstone bottom. The rear consists of a concrete platform supported on longitudinal beams and circular steel piles and carrying the traversing rails of the loading tower. The two sections are independent and the tower line is thus fully protected by the front section from any accidents on mooring.

Depth alongside is at present 12·5 m at low spring tides with a tidal range of 2 m, sufficient for a 45 000 tdw boat. The structure has been designed for a final depth of 15·5 m, sufficient for a 100 000 tdw boat. This can be achieved with very little dredging.

Workshops and power-house

Near the port are situated the railway workshops, a small workshop for sea and conveyor maintenance, the stores and offices, an 8000-kVA diesel power-house cooled by sea water, and 5000 m^3 of fuel storage put in by a supplier under contract to Miferma.

Town

The town, 'Cansado', is laid out 5 km north of the port on a promontory within the bay, protected from coarse sand blown at ground level by a wide expanse of water. It comprises 32 'senior staff', 148 'foremen's', 470 'workmen's' houses and 60 bachelors' flats. Layout and amenities follow the same principles as at Zouerate, but are more ample. Air-conditioning is not necessary because of the trade winds, but the layout had to take into consideration some blown sand which tends to build up into dunes against the buildings.

All roads are tarmac and a complete telephone system covers both town and port. A permanent wireless link connects Cansado to Zouerate.

PLANNING AND CARRYING OUT THE PROJECT

Early difficulties were magnified by the lack of a background organization which could be called upon when required, such as exists in old-established companies. This was particularly felt in 1957 when the final detailed information had to be obtained and the technical and financial programme laid out.

Key personnel were provided through the shareholders and through the French Government. They carried out the planning and design and later supervised the work. Fifteen consultant firms, specialists in various branches of civil, mechanical and electrical engineering, assisted them. Various eminent specialists were also consulted on specific aspects of the project—geology of the orebodies, railways in sand areas, etc.

The company relied heavily on consultants as a matter of policy in order to reduce the number of personnel who would become redundant when planning changed to construction and construction to operation.

The shortage of adequate mining personnel was overcome by a contract with Société Minière et Métallurgique de Peñarroya who undertook to provide the staff and organization necessary to plan and develop the mine and to run it during the first years of operation.

As the project took shape recruitment became easier. The staff were divided into three departments: 'pilot', 'operative' and 'planning'.

The 'pilots' designed the plant, chose the equipment, negotiated with suppliers, ensured that construction followed design, recruited final staff, prepared operating instructions and organized operation. Separate pilot departments handled mine, railway and port.

The 'operatives' translated the design into constructional terms, estimated the logistics and negotiated with contractors. Once a contract had been signed the engineer responsible went out on site and controlled the work. A single operative department covered the whole project.

The 'planners' co-ordinated the logistics and the individual activities into a coherent whole as regards design, construction and timing.

The project was broken down into eight 'jobs' and 29 'constructional contracts'. 'Jobs' corresponded to operational units. Pilots and planners could thus follow each component part of the project and ensure that it was completed satisfactorily and in a logical sequence. 'Constructional contracts' corresponded to constructional units which could be entrusted to a single contractor. They overlapped part or all of several jobs and were planned so that 'jobs' could be completed on time and in the desired sequence and so that the contractor could make the best use of his men and equipment.

A further 28 contracts for the supply of major equipment were handled by the 'pilots'.

The various contractors subsequently grouped themselves and the civil, mechanical and electrical engineering was handled by eight major contractors working with eight subcontractors.

The two organizational charts (Figs. 10 and 11) give the distribution of key personnel during the planning period and the construction period.

The total number of personnel employed on the project is difficult to estimate because of the heavy call made on the consultants. The number of employees for Miferma and its subsidiaries, between Paris and Mauritania, reached 535 Europeans and 1370 Africans. Peak contractor's personnel reached 500 Europeans and 3000 Africans.

Each 'job' was laid out on a Gantt chart and the details analysed as regards type and volume of work and timing. As the starting date had not yet been decided, time was counted in negative months, 'M^0' being the month of the first shipment of ore. The above details were then reassembled on a parallel series of charts according to the similar nature of constructional work. These were then analysed as regards the volume of work and its distribution in time and space and the findings

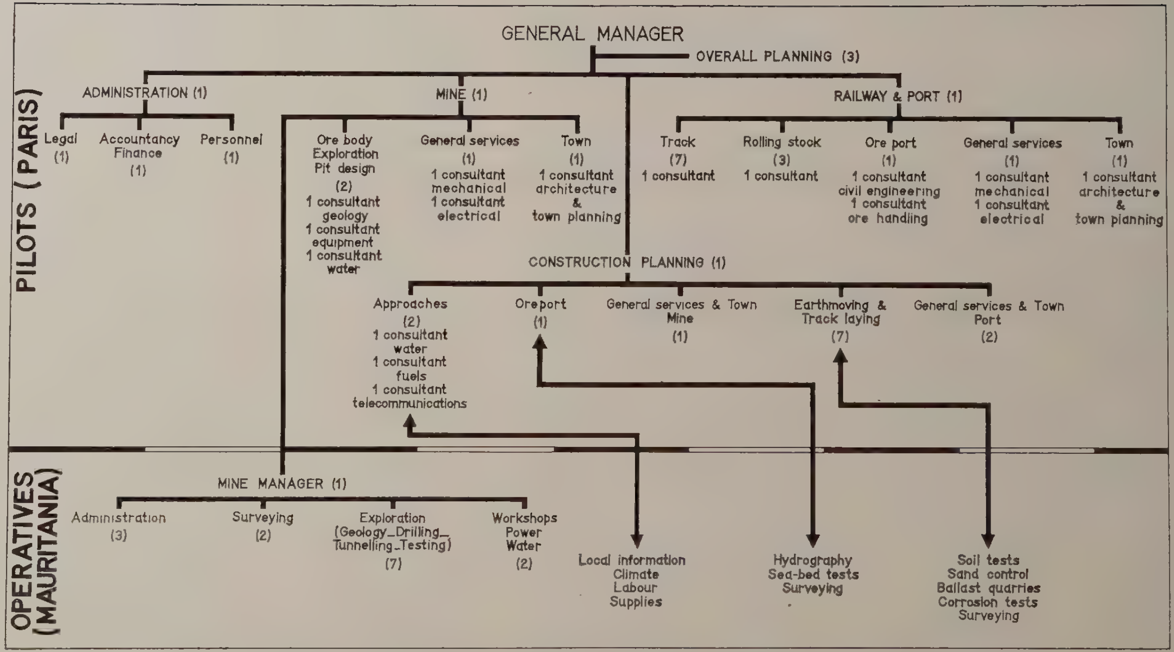

GENERAL MANAGER
OVERALL PLANNING (3)
PILOTS (PARIS)
ADMINISTRATION (1)
Legal (1)
Accountancy Finance (1)
Personnel (1)
MINE (1)
Ore body Exploration Pit design (2)
1 consultant geology
1 consultant equipment
1 consultant water
General services (1)
1 consultant mechanical
1 consultant electrical
Town (1)
1 consultant architecture & town planning
RAILWAY & PORT (1)
Track (7)
1 consultant
Rolling stock (3)
1 consultant
Ore port (1)
1 consultant civil engineering
1 consultant ore handling
General services (1)
1 consultant mechanical
1 consultant electrical
Town (1)
1 consultant architecture & town planning
CONSTRUCTION PLANNING (1)
Approaches (2)
1 consultant water
1 consultant fuels
1 consultant telecommunications
Oreport (1)
General services & Town Mine (1)
Earthmoving & Track laying (7)
General services & Town Port (2)
OPERATIVES (MAURITANIA)
MINE MANAGER (1)
Administration (3)
Surveying (2)
Exploration (Geology_Drilling_Tunnelling_Testing) (7)
Workshops Power Water (2)
Local information Climate Labour Supplies
Hydrography Sea-bed tests Surveying
Soil tests Sand control Ballast quarries Corrosion tests Surveying

Fig. 10.—Miferma: organizational chart of key personnel during the planning period. (Figures in brackets represent the number of responsible men employed.)

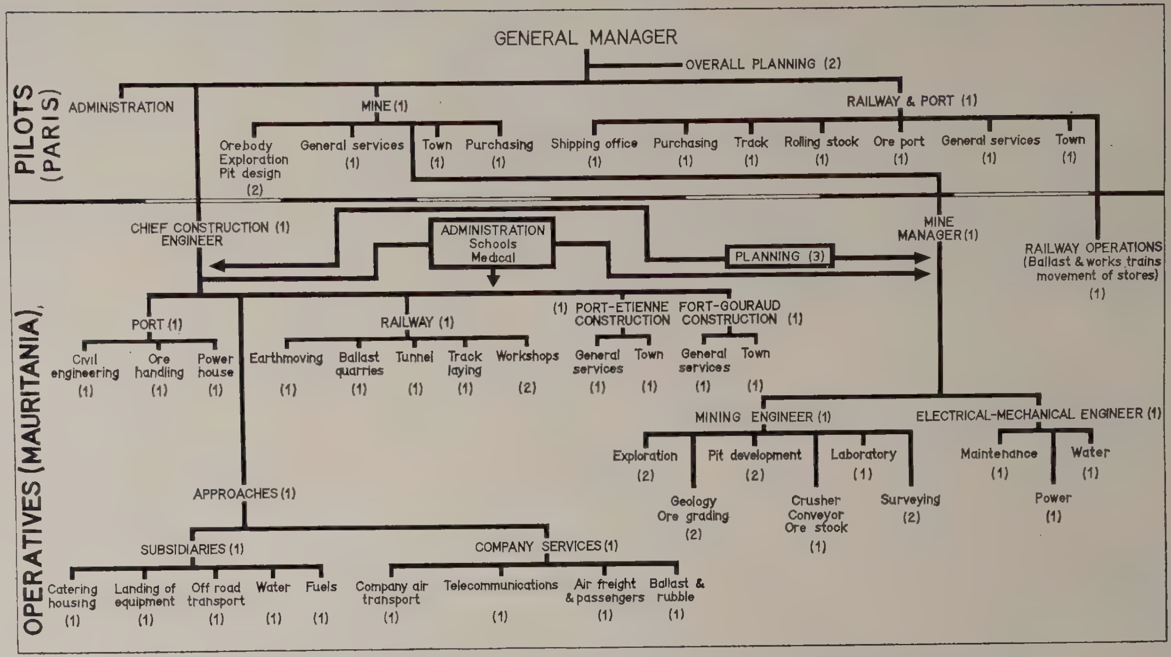
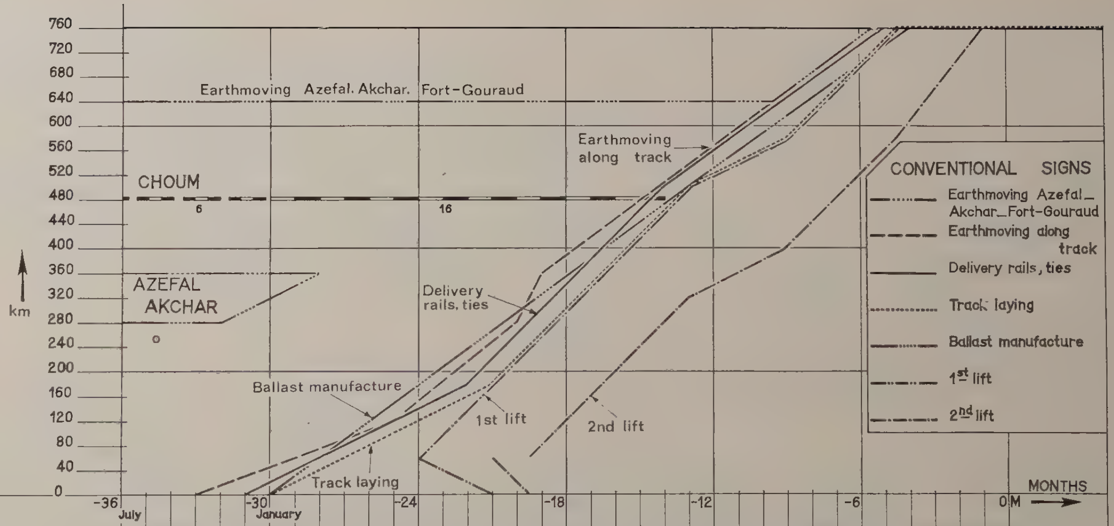


Fig. 11.—Miferma: organizational chart of key personnel during the construction period. (Figures in brackets represent the number of responsible men employed.)



	-36 July						-30 January						-24						-18						-12						-6					
Earthmoving in km				15	20 15	20 15	20 15	20 15	15	15	15	30	30	30	30	30	80	30	30	30	30	30	30	30	30	30	30	15 15	30	30	30	10				
Delivery of track, equipment in 000 tons						6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	10	10	10	10	10	10	10	10	10	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5					
Track laying in km							20	20	20	20	20	20	20	20	20	40	40	40	40	40	40	40	40	20	20	20	20	40	40	40	40	20				
Ballast manufactured in 000 m^3							23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23				
1st lift in km													60	60	60	40	40	40	40	40	40	40	40	20	20	20	20	40	40	40	40	20				
2nd lift in km																40	40	40	40	40	40	40	40	20	20	20	20	40	40	40	40	60	70	70		
Locomotives											3	3	1																							
Ballast cars											30	32																								
Sundry cars											30	30	15																							
Ore cars (equipped for ballasting)												12																								

Fig. 12.—Planning of the railway construction.

were assembled on one or several charts each corresponding to a separate homogeneous contract. Care was taken to ensure that the timing of each item in one contract was such that it did not hinder other work in other contracts; also that it might be, if possible, of assistance in the prosecution of such work.

Each contract was then laid out on a two-dimensional chart (Fig. 12) which showed the progress of work as a curve with the quantities as ordinates and times as abscissae.

A parallel series of charts covered the time for calls and reception of tenders, choice of contractors or suppliers, manufacture and delivery on site. Along the time scale was marked the progressive expenditure on the contract.

A series of Gantt charts reassembled the individual sectors of each contract into the original jobs.

Similar charts were prepared for the logistical problems, tonnages and nature of goods to be carried by land, sea or air.

The overall initial timing of the project is given in a simplified form on the Gantt chart (Fig. 13).

These charts, kept up to date, not only served for planning but also during construction. The actual rates of progress and expenditure could be plotted against the forecasts and the trend of the curves gave good warning of the progress of work. This control enabled work to be speeded up in the middle of the operation and gained several months on the original timing.

This detailed analysis showed up a series of local problems which had to be solved before contractors could be expected to submit valid tenders and which could not be left to the discretion of the contractors without creating bottlenecks, viz. Port Etienne was the only possible landing point; it had to be developed from a capacity of 2000 t/year to one of 24 000 t/month; Port Etienne had no communications with the hinterland; there was a general lack of roads and airfields, no social or industrial infrastructure, lack of suitable building materials including good stone and sand, lack of water, lack of food resources, lack of housing, lack of telecommunications, lack of skilled labour, shortage of unskilled labour, and heat, drought and dust were wearing on men and equipment.

It was decided that the company would provide the corresponding services and charge them at a fixed rate to the contractor, thus enabling the latter to base their estimates on known unit costs. This involved planning and costing all requirements in the same detail as the actual operation and setting up organizations capable of coping with them.

Such services as would still be required by the company during the operational period were covered by special departments to provide a shipping office, telecommunications, air transport, base maintenance,

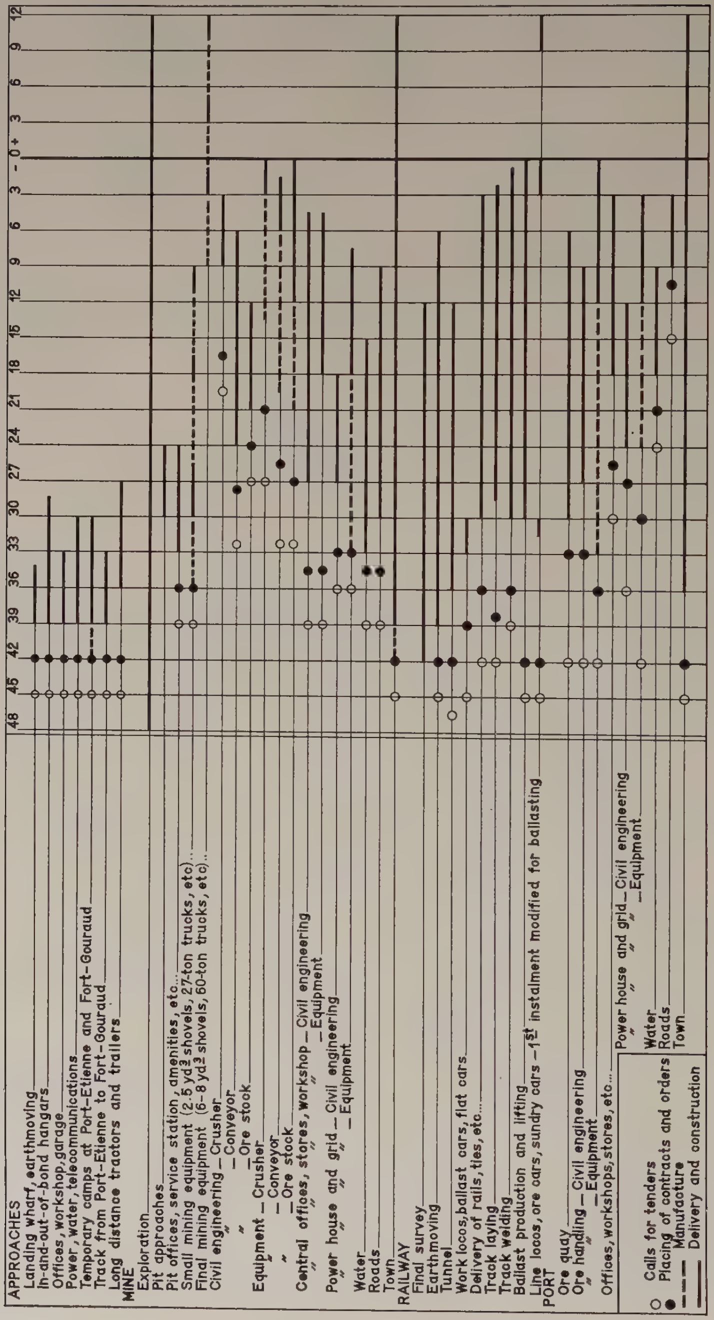


Fig. 13.—Progress chart of the Miferma development.

medical services, bulk food storage, power at the two terminals and rail transport as the line advanced.

Services of a temporary nature requiring a highly specialized organization were covered by creating subsidiaries with companies already experienced in such work. These covered landing and warehousing, land transport, water supply and development, and terminal camps. Miferma provided the basic equipment and capital, operating costs were maintained at the target estimate submitted to the contractors and provision was made for the subsidiary to operate in Mauritania as an independent concern once the construction period was over.

The operation of these subsidiaries and departments was co-ordinated by a special department.

While the above preparations were going on in Paris, field teams were collecting information and preparing the ground.

At the port the sea bed and the ground were surveyed and tested, meteorological data collected, sand control experimented and areas, protected from blown sand, chosen and pegged out for the town and the industrial area.

On the railway, three sites were chosen for ballast quarries along the line, a ribbon 400 m wide was surveyed in detail and pegged along the proposed centre line, the tunnel area was surveyed and drilled, tests were carried out on the behaviour of cuts and fills, the behaviour of the moving dunes was studied and methods devised for their destruction or stabilization. Sections of rail were laid out at different heights from the ground and at different angles to the direction of the dominant winds to study the build-up of sand and rail temperatures registered throughout the 24 hours.

In order to complete the picture on the mine in time to be of use for pit planning, as soon as money became available early in 1960 prefabricated houses were put up at the mine, the staff was increased, the workshops were extended and 1100 kW of diesel power installed, a small laboratory was equipped and the road to Tazadit improved.

A $2\frac{1}{2}$-yd^3 shovel, two 27-ton trucks, two bulldozers, a light crawler-mounted percussion drill, a down-the-hole drill, designed for exploratory work and deep holes with a range of 120 m, and a 12-in self-propelled tricone drill were brought in to Tazadit.

Benching was started immediately and the first cuts were not devoid of difficulty. The ore was dumped into a nearby ravine not far from the crusher site, where it could easily be reclaimed.

The benches were started in ore from a low point on the crest (710 m), which was first levelled off to the south. The shovel then turned northwards and, as space became available, a further three $2\frac{1}{2}$-yd^3 shovels, sixteen 27-ton trucks and several bulldozers and drills were brought in.

The benches were extended to cover the whole orebody, while two shovels attacked the 'bun' of BHQ. All blast-holes were sampled and assayed for Fe and some for SiO_2 and Al_2O_3.

This small equipment will ultimately be used on F'Derik.

At the same time the down-the-hole drill and the tricone were put to drilling a systematic grid at 50-m intervals, to an average depth of 60 m. Closer spacing was impracticable until the ground had been levelled off. Samples were taken continuously in 3-m lengths. All were assayed for Fe, and all samples above 58 per cent Fe, and some samples of low-grade material, were assayed for SiO_2, Al_2O_3, P, S, TiO_2 and H_2O loss at 105°. Some samples were also assayed for Mn, BaO, CaO, MgO, Ni, Co, Cu, As and Pb.

A mechanical screening and sampling plant was installed and 2 per cent of the shovel output was screened at 100 mm, 40 mm, 6·5 mm and 2 mm. Each fraction was weighed and assayed.

Small pits were dug in the benches, the spoil was carefully weighed and the volume of the pits measured by pouring in fine dune sand from a graduated 40-gal drum to measure the specific gravity *in situ*.

Two powerful diamond drills were purchased and a series of inclined holes put down to explore the 300-m level. The first two holes, now completed, show a horizontal thickness of 100 m of hard high-grade ore at that level. The remaining holes must be completed before pit engineering to this level is carried out.

All results were shown as contents curves on plans and sections and plotted on statistical charts.

This work was fruitful:

(*a*) The behaviour of the various materials was studied in the faces.
(*b*) Mining results were compared with those forecast by exploration and by systematic drilling, and selectivity assessed.
(*c*) The distribution of the components was studied according to grade, grain size and position in the orebody.
(*d*) Data were obtained on the behaviour and maintenance of equipment prior to ordering.
(*e*) Drilling and blasting data were available before orders were placed.
(*f*) Benches were prepared to take the big shovels.
(*g*) The pit was designed for the first 15 000 000 tons of ore.
(*h*) A 1 500 000-ton stock of ore was built up to supplement output in the initial years.
(*i*) Data were obtained on dust control, road maintenance, angles of repose, etc.
(*j*) Methods of forecasting, ore grading and blending were adjusted.
(*k*) The study of ore beneficiation was initiated.

At the time of writing (January 1964), some 2 800 000 tons of ore, 800 000 tons of off-grade 'beneficiating' material and 4 600 000 tons of waste have been taken from the pit.

The operation is proceeding according to plan with particular attention to problems of programming and ore grading based on forward drilling.

The laboratory is being expanded and the use of X-ray spectrometry is being investigated.

While the above work on the pit was in progress, other teams were driving the conveyor tunnel and crusher pit, and contractors were putting in the pit offices and workshops, the power line, the conveyor line and ore storage and, finally, the crusher.

F'DERIK

At F'Derik exploration results were far more difficult to interpret and it was finally decided to put in a longitudinal drift at the 420-m level carrying inclined lateral diamond drill holes 100 m long, and 40-m drifter holes every 20 m. All holes were fully sampled.

Surface drilling and trenching were also continued.

The surface section and internal section thus established, it was possible to assemble the vertical drill sections and show up the structure and distribution of the first 10 000 000 tons to be mined.

WORLD BANK APPROVAL

The above operations were followed by the World Bank to ensure that work complied with their standards and that the specifications followed the strict procedure laid down in their regulations.

All principal specifications were submitted to the Bank to ensure that they were truly international. Calls for tenders were then publicized in the Press and through the Embassies.

The tenders were then opened by a jury and a report prepared setting out the details of the bidding and justifying the choice of contractors or suppliers. This was submitted to a 'contracts committee' representing the shareholders of Miferma and then sent to the World Bank for approval before being notified to the contractor.

THE CONSTRUCTIONAL PERIOD (1960–63)

At the time the World Bank granted the loan the construction operation had been planned in detail, the contracts for preliminary work were ready for signing, and the time of completion had been set at 42 months. The first teams landed at Port Etienne on 10th April 1960.

Installations at port and mine

A sheet-pile landing wharf was built along Port Etienne beach. The ground behind it was levelled off and filled, and in- and out-of-bond hangars were put up. Tugs, lighters, a 100-ton floating crane and handling equipment were brought in and handed over to the operating subsidiary. A large railway and general workshop was erected. An old oil tanker, berthed off the existing fishing pier, provided temporary fuel storage. A temporary camp of aluminium huts and prefabricated communal buildings was handed over to be run by a subsidiary. Prefabricated offices and a medical–surgical unit were installed.

Most of the prefabricated units were built in sections 10 m long, shaped as railway carriages, which could be lifted on to a flat truck for transport. Each was thermally insulated, air-conditioned and equipped as living quarters, mess hall, kitchen, sanitary unit, etc. For buildings requiring large floor areas one or both sides of the component units were taken off upon arrival at site and the units bolted together with a rubber sealing joint. These units were used on the inland sites and a camp could be moved and reassembled within a day.

Completion of permanent dwellings and industrial buildings was timed to correspond with the increases in staff and requirements. By mid-1962 the temporary camps were no longer in service.

Construction at the mine followed lines very similar to those at Port Etienne. Great care had to be taken as regards logistics and stores as the mine was at the end of the receiving line and very sensitive to any disturbance in supplies.

Water supply

In the first days water was supplied either by a small sea-water distilling plant which already existed at Port Etienne or was brought in by tanker from Dakar. Meanwhile, three bore-holes were sunk at Boulanouar and equipped for pumping. Water was brought in first by truck and then by train and this is still the method employed.

Communications

Five small inspection and personnel planes were brought in as well as a fleet of four-wheel-drive personnel cars.

All inland sites including the mine, all planes and vehicles, and all construction sites at port were connected by wireless to a central station at the Port Etienne camp.

The Port Etienne air strip was extended to take DC6s; that at Fort Gouraud to take DC4s; and small airstrips were levelled off along the railway line and at Zouerate.

As soon as the landing installations were completed, earthmoving began on the railway line from the landing wharf to the port site, to the first ballast quarry situated 60 km inland and to the water point of Boulanouar, 40 km farther on. Rails were laid by hand directly on the subgrade while the rail welding plant was being put up at Port Etienne.

Simultaneously, a track, capable of taking heavy traffic, was bulldozed through to the mine along the railway route and a good road, subsequently tarmacked, was laid between the landing wharf, the camp and the port and town sites. Sixteen road tractors and twenty-three 35-ton trailers were brought in and handed over to the operating subsidiary.

On the railway a team of surveyors provided by the company and by the contractors laid out and marked the final centre line and agreed the volume of earthmoving. Twenty km behind them the earthmoving gang levelled off the subgrade by bulldozer and scraper.

When the railway line reached the first ballast quarry and this had been equipped, the fleet of ballast cars was ready, as well as a number of ore cars temporarily modified to spread ballast. Distances were still short and the track could therefore be normally ballasted as it advanced and simultaneously ballasted back to Port Etienne. Undersize from the ballast was used as rubble and sand for concrete.

The track-laying train consisted of two-line locomotives with, ahead of them, a series of bogies carrying 34 'rails' each welded into 144-m lengths and, behind them, a series of flat cars loaded with bundles of ties. The ties were brought ahead of the train by a truck running parallel to the track and distributed along the subgrade by a crane. The bundles were then opened up and the ties distributed by hand along the subgrade according to a wire scale laid parallel to the centre line. Two rails were then hitched behind a tractor running on the subgrade and straddling the ties and pulled into position through roller guides. The rails were immediately bolted down and the train moved forward to the end of the track.

The ballast trains dumped their load immediately behind the work train. This was tamped by portable mechanical tampers and the rail bolts tightened in order to clear and stabilize the line.

Two km to the rear, three automotive tampers and levellers bolted the track into its final position and 2 km beyond this a small gang gave the final adjustments to the track.

By 1962 the rate of advance had reached 2000 m a day.

Twenty kilometres behind the track-laying gangs, the 144-m lengths were aluminothermally welded into a continuous rail between crossing points. Welding was kept entirely independent of laying as it could only be carried out during certain hours of the day and had to be suspended when conditions were unsuitable.

The railway camps were built on temporary spurs and moved forward every 30 km. The quarry equipment was moved forward from one quarry to the next to remain as close as possible to the site.

By the time the line reached the Choum tunnel this was ready to receive it.

A tarmac road 10 m wide was laid from F'Derik to Zouerate and to Tazadit and a pipeline from the pumping areas to the town and to the pit.

Transport during the constructional period

Supplies and water were brought to the working sites and to the mine in the 35-ton trailers. They covered a total of 2 500 000 km and carried 85 000 tons over various distances.

As the line moved forward, the trailer base camp moved forward with it and supplies were brought as far as possible by train.

Very heavy items of equipment, such as crusher parts, were brought in by road on a special 16-wheel low-slung trailer. The sites and equipment were ready to receive them and six months later the ore began moving to the port.

The first ore boat was loaded on 15th June 1963—six months ahead of schedule.

622.1:526.918.52:622.013.36 *Paper 21*

Determination of Volumes in Opencast Mining by Aerial Surveys

R. F. RAWIEL, Dipl. Ing.

Technical Manager, Aircraft Operating Co. (Aerial Surveys), Ltd.,
Johannesburg, South Africa

SYNOPSIS

Photogrammetric methods can be employed to advantage for the measurement of volumes in the mining industry. Aerial surveys can be applied to the volumetric assessment of stockpiles, dumps, excavations, overburden and the capacity of potential dam sites. The size of the survey must warrant the employment of photogrammetric methods. If a certain area has to be surveyed periodically, aerial surveys can also be employed economically. Volumetric assessment from aerial surveys can yield the same accuracy as volumetric assessment by conventional methods.

Two examples illustrating the application of aerial survey methods are given, one to reclamation work in the area of a defunct mine, and the other to volumetric assessment in an opencast mine.

Disadvantages of the photogrammetric methods are discussed. Data from photogrammetric procedures can, however, be processed in the same way as those from conventional surveys, and there is the advantage that the stereo plotting instrument can supply data directly in a form suitable for electronic computers.

In recent years photogrammetric surveys have played a vital role in providing maps and plans for various purposes. Large development projects in many spheres could have been seriously hampered without the availability of accurate maps and plans at comparatively short notice. One of the outstanding advantages of aerial surveys is the fact that large survey projects can be carried out much more quickly than conventional surveys. While the general conception of photogrammetric surveys differs largely from conventional survey practice, photogrammetric surveys must, nevertheless, be considered as a coherent part of the general survey practice and not as a procedure in its own right. The theory of aerial photogrammetry is well covered in a number of excellent studies[1,2,3] and it would go beyond the scope of the present paper to give a lengthy explanation of the basic principles involved. The more specific characteristics of aerial photogrammetry as applied for the purpose of volume determination may, however, be outlined.

Aerial photogrammetry used to be accepted, for all practical purposes, as a procedure for providing maps and plans only. The perfection of the photogrammetric instruments, the refinement of methods and the availability of the electronic computer have advanced aerial surveys one step from the former purely graphic presentation of results. Aerial photographs, taken with modern, high-precision cameras, can provide much more useful information than can be accommodated even on a very large-scale map. Furthermore, the accuracy with which data can be extracted from an aerial photograph exceeds the plotting accuracy on a large-scale map. To utilize adequately the full potential of aerial photogrammetry, information from aerial photographs can be presented in numerical form. In most cases the numerical data required are the co-ordinates of specific points (position and height). Other details not usually incorporated in a map are cross-section or profile data, either presented directly in the form of a drawn curve, or numerically, being read from machine counters. Some photogrammetric instruments are equipped to draw profiles directly from the stereo model as established within them. These profiles can be drawn at different horizontal and vertical scales in any direction and from any given point on the stereo model.

The main problem in the compilation of maps from aerial photographs is the transformation of the perspective photograph into the orthogonal projection of a map. This transformation is achieved (*a*) by the restitution of an apparent stereo model from two overlapping photographs, and (*b*) by the subsequent evaluation of this stereo model by a mechanism allowing the orthogonal projection of the information on the stereo model on to the map sheet.

The stereo model, as restituted from the overlapping aerial photographs, is a geometrically true replica of the terrain photographed. A floating dot mark is employed to scan the model and a pencil point follows truthfully every movement of the dot mark, tracing the required map on to a drawing-board. The relative height of the floating mark above a given datum can be controlled and adjusted in that topographical detail of the model can be traced merely by following its course or outline while constantly adjusting the height of the dot mark to keep it in touch with the model surface. Contours are drawn by setting the floating mark to a fixed height, i.e. the height of the required contour, and by probing the dot mark exactly along the surface of the model. Spot-heights can be read from the stereo model by setting the floating mark on the surface of the model at the required spot and reading its height from an instrument counter.

Aerial photographs for mapping purposes are taken with highly specialized cameras having practically distortion-free lenses. The picture size of the camera types most commonly used is 9 in. × 9 in. and the focal length of the lens is 6 in. To achieve the necessary stereoscopic overlap along a flight strip, each photograph within the strip overlaps the preceding and the following photographs by 60 per cent. The side overlap between flight strips amounts normally to 25 per cent.

To establish the scale and the height datum for the final map at least three ground reference points have to be provided for each pair of stereoscopic overlapping photographs. These points can take the form of precisely identifiable topographic detail or specific points, clearly marked on the ground prior to photography. The position of such points, related to a common survey system in planimetry and height, must generally be established by ground survey methods.

The final product—the map—is derived from conventional surveys as the interpolation and the connection of a number of surveyed points. The photogrammetric map is drawn directly from the stereo model, as established in the stereo plotting instruments. In this way the photogrammetric map can present considerably more detail—and variations of topographic features and contours—than the conventional map. On the other hand, it must be appreciated that conventional surveys are carried out on the landscape itself, i.e. on a model at the scale 1:1, whereas photogrammetric maps are drawn on a small-scale model of the terrain, assuming that this stereo model is geometrically true. Several prerequisites are necessary to establish a geometrically true model:

(*a*) the aerial photographs must be taken with a camera that guarantees distortion-free photographs or with a camera of known distortion, for which allowance can be made;
(*b*) the base of the photographic emulsion must be stable, and may not shrink or expand differently in any direction;
(*c*) the stereo plotting instrument must be designed to enable the correct geometrical restitution of the stereo model, and its mechanical deficiencies must be of a lower order than the pointing accuracy, which is governed by the resolution of the photographic image;
(*d*) the necessary ground reference or control points must be identifiable with sufficient accuracy and must be surveyed to a compatible degree of accuracy, regarding the scale of the stereo model; and
(*e*) the efficiency and the aptitude of the photogrammetrist must be such that the potential efficiency of the instrument can be fully realized.

It is clear that none of the above conditions can be fulfilled completely and any variation in the final result from the anticipated optimum must be considered as an accumulation of errors from all the above sources. It is also obvious that the magnitude of any of the allowable accuracy tolerances must be in sound proportions to each other. The gain in accuracy by the high resolving power of the camera lens would be lost by a coarse grain of the photographic emulsion or by excessive mechanical deficiencies of the stereo plotting instrument. An inefficient operator would not, of course, be able to make full use of the machinery put at his disposal. Practical experience and the refinement of the instrumentation have brought aerial surveys to a state where results can be guaranteed unconditionally to predetermined degrees of accuracy. The photogrammetric method has the potential for portraying topographical detail, and especially contours, more faithfully than the conventional method. It does not, however, follow that the photogrammetric method is in itself more accurate. More than with conventional methods its accuracy depends on the available resources and routines will have to be specifically adjusted to the purpose of a survey project. Costs for aerial surveys rise steeply with increased demand for accuracy and there is a definite limit to which aerial surveys can be employed economically—mainly due to the high investment cost and the complex organization of a photogrammetric enterprise.

A number of factors determine the accuracy of photogrammetric surveys: primarily, the scale of the stereo model as established in the stereo plotting instruments governs the accuracy and the extent of information extracted from the aerial photograph. The larger the scale of the stereo model, the larger will be the scale of the aerial photography and, consequently, the smaller the area covered by each stereo model. Where a high degree of accuracy is required each model has to be controlled by at least three ground control points. The cost of ground control will therefore increase considerably with the number of stereo models required to cover a given area. Naturally, the type of stereo plotting instrument used has a bearing on the accuracy of the final product. Instruments employed for high-accuracy standards are very costly and the question of depreciation has to be taken into consideration. The extent of the survey project and the time factor involved, however, in most cases determine whether photogrammetric methods should be employed. The economic advantages of aerial survey can only be realized where the size of the area and the time limit warrant its employment.

VOLUME DETERMINATION BY PHOTOGRAMMETRIC METHODS

The actual data derived from photogrammetric procedures for volume determination are largely the same as provided by conventional methods—(*a*) contours; (*b*) sections; and (*c*) random spot-heights.

The further processing of these data can be treated in exactly the same way in both cases, with the possible exception that numerical data as received from the stereo plotting instrument can be directly punched on tape or cards, without transcription by hand, and fed into electronic computers for evaluation.

Contours

The greatest advantage of the photogrammetric method is the facility of plotting contour lines directly on to the final map without having to derive them from interpolation or as joins between surveyed points. If volumes are determined from measured areas between contour lines this advantage can be fully utilized. The method should be employed where surfaces are irregular, as in opencast mining, where periodic assessment of the excavations must be carried out, for the determination of overburden or for the capacity of potential dam sites. The contour interval may be selected to suit the particular object. It is normal practice to guarantee the contour accuracy to be within half the contour interval, i.e. a 5-ft contour line should not deviate more than 2·5 ft on either side from the true value. The scale of photography will be selected to guarantee the specified contour accuracy and, at the same time, to cover an area as large as possible. High-precision cameras, in conjunction with first-order plotting instruments, can yield a height accuracy of approximately 0·2 ft per 1000 ft of the flying height. Therefore if a height accuracy of 0·6 ft is required, a flying height of 3000 ft above ground level would have to be selected. At this flying height one stereoscopic overlap covers 3375 ft × 1800 ft (140 acres). It is not advisable to select a closer contour interval than the accuracy of the photography allows.

Sections

Under normal circumstances sections will be derived from spot-height measurements along a straight line. This method can also be applied in photogrammetric measurements of sections. The first-order stereo plotting machines are equipped with reading devices (counters) which allow the numerical read-out of X, Y and Z co-ordinates, based on the internal machine system. Obviously, for cross-sections only the Z co-ordinate (height) is of interest and can be read from the Z counter, whereas distances between selected points along the section are normally plotted on the drawing-board. A number of stereo plotting machines are equipped with special read-out devices which record the three co-ordinates or any one of them in clear print on a typewriter and if required, are punched on tape or cards. The accuracy of the individual spot-heights can be assumed to be slightly higher than the contour drawn from the same stereo model—understandable if one considers that individual spot-heights can be read more than once and a final mean can be taken.

One further characteristic of the photogrammetric survey which must be stressed is the fact that the number of spot-heights read on any one model does not influence the economy of the survey itself materially. The photogrammetrist is able to select as many spot-heights as he considers to be necessary to portray any section truthfully. The fact that he has a scaled model at his disposal to select the necessary cross-sections in detail, without the necessity of pacing the ground in order to find the most suitable sections, enables him to carry out the work more efficiently and in clearer perspective.

Auxiliary equipment, to be used in conjunction with the first-order plotting machines, has been developed to draw profiles and cross-sections directly on the drawing-board. The horizontal and vertical scales for these sections can be selected differently. The final graphical representation is a curved line—the terrain surface between given points referenced to a straight line. The floating dot mark has to be guided along a straight line (the sections), while the height of the dot mark is adjusted exactly to the height of the terrain along the selected line.

Random Spot-heights

Apart from spot-heights as observed within the section, random spot-heights can serve a useful purpose for the determination of volumes. The stereo model provides a convenient means of selecting and measuring any spot-height necessary to determine volumes more accurately.

Conventional surveys normally can provide a greater accuracy in point determination. Aerial surveys, on the other hand, allow a better selection of points and more points can be measured with very little extra effort. A greater point accuracy than ± 3 in. is seldom required for ordinary volume measurements where irregular shapes are encountered. More important in the accurate assessment of volumes is the selection of points to be measured. Here aerial photogrammetry has a clear advantage over conventional methods and, in order to achieve the same accuracy tolerance as in ground survey methods, a slightly inferior point accuracy in aerial survey methods can be compensated by better selection and a greater number of points.

DISADVANTAGES OF PHOTOGRAMMETRIC METHODS

The fact that all aerial photographs picture the surface in a perspective projection has already been mentioned. The surface, covered by one photograph, is pictured through the lens of the camera as the centre of projection. The type of lens most commonly used is the so-called wide-angle lens, covering an angular field of about 90°, so that the area away from the centre of the photograph can be seen only from an oblique angle. Occasionally, therefore, it happens that vertical or steep parts of the surface are obscured if these parts are pictured near to the edge of the photograph, facing away from the centre.

This must be considered in planning the photography, in particular for opencast mining, and sometimes additional photographs will have to be taken to cover an area adequately. Clearly, in some instances the workings in opencast mining may be such that aerial photographs cannot provide a full coverage at all.

Another obstacle, which again occurs particularly in the case of opencast mining, is the interference of shadows. This problem can be largely overcome by using special film and developer and by subsequently producing prints from that film by an electronic printer, where the exposure time of minute areas of the negative is individually adjusted to its density.

Other peculiarities of aerial photogrammetry, mainly the economic factor, have already been dealt with, but it should again be mentioned that aerial survey can only be economically employed where the extent of the survey and the time limit warrant it.

PROCESSING OF DATA

All data received from photogrammetric procedures can be brought to exactly the same form as the data received from conventional surveys. The further processing of data can then follow the same pattern in both cases. One advantage peculiar to the photogrammetric process is the fact that the stereo plotting instrument can supply data directly in a form suitable for electronic computers. Many stereo plotting instruments are supplemented by automatic recording devices furnishing digital data, such as X, Y and Z machine co-ordinates, or any one of them, in clear print and in the form of punched information on tape, or on cards. If the punch is coded to the specific requirements of an electronic computer the tape or the cards can be processed by the computer without any manual transcription, thus ensuring greater economy and avoiding all transcribing errors. How far electronic computers can be employed economically for the computation of volumes depends on individual requirements. It must also be appreciated that the use of the electronic computer entails laborious programming and the employment of a complex organization.

In general, electronic computers should only be employed where the same routine calculations have to be executed in great numbers or where reiteration of the same calculations with different parameters must be calculated.

PRACTICAL APPLICATIONS OF AERIAL SURVEY TO VOLUMETRIC ASSESSMENT

The following example demonstrates the application of aerial survey methods for volumetric assessment.

The terrain of a defunct mine had to be reclaimed for industrial purposes and the object of the survey was to establish the level to which

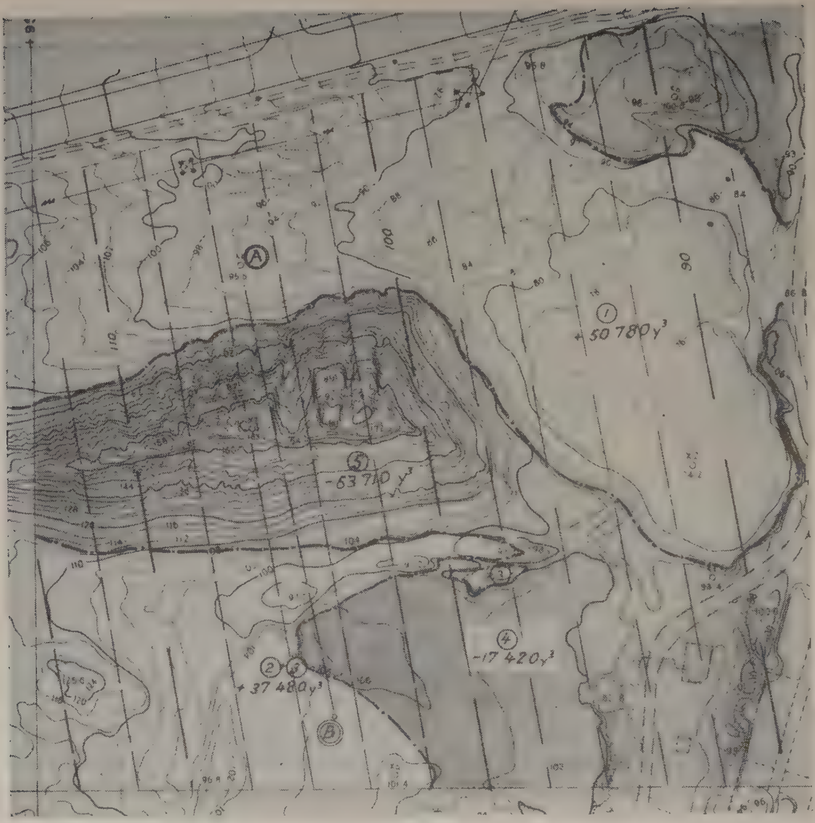


Fig. 1

the ground would have to be raised if the various waste dumps were dispersed over the mine area.

A photogrammetric map was prepared from aerial photographs taken at a height of 3000 ft above ground level with a wide-angle camera (Aviogon type lens). The contour interval of 2 ft was chosen to present the shape of the dump and the surrounding areas adequately.

Survey marks, previously established for mining purposes, were signalled by white crosses and were used to control the stereo model. The map was compiled on a first-order stereo plotting instrument.

Fig. 1 shows a section of the final map; the shaded area indicates the waste which has to be moved to fill the blank areas. All volumes were derived from measuring the areas between contour lines and the application of the prismoidal formula. The total amount of waste to be moved on the project was approximately 700 000 yd^3, estimated to an anticipated accuracy of approximately ±2 per cent.

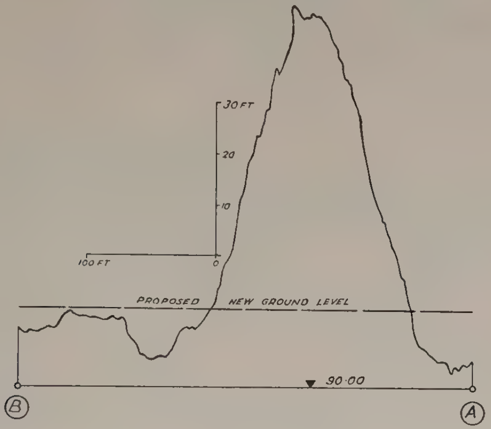


Fig. 2

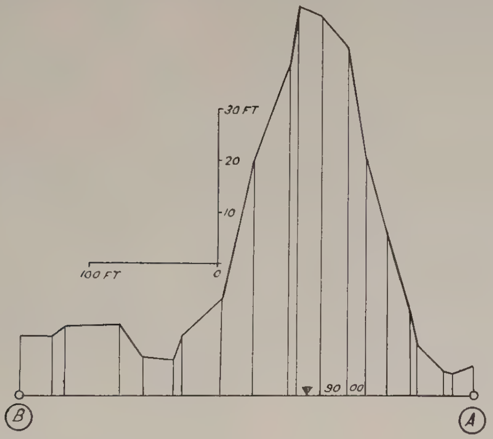


Fig. 3

Fig. 2 shows a cross-section from *A* to *B* on Fig 1, through the waste dump, drawn directly on the stereo plotting instrument. Fig. 3 shows the same cross-section plotted from spot readings on the instrument.

As has already been explained, the spot readings yield a greater accuracy at the selected point, but the continuous line drawing of the cross-section does result in a more faithful portrait of the surface.

The second example (Fig. 4) illustrates the volumetric assessment in an opencast mine. Photographs were taken at 3000 ft above ground level and the contour interval selected was 4 ft. Each set of contours was plotted separately in order to exclude the otherwise unavoidable

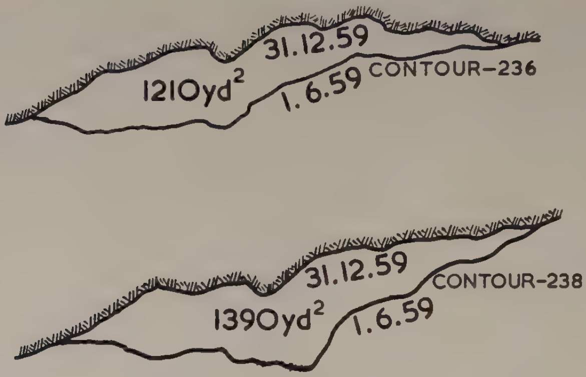


Fig. 4

overlapping of contours. The area between corresponding contour lines plotted at six months' time interval is measured with the planimeter and the volume of quantities removed is determined, applying the prismoidal formula.

REFERENCES

1. *Manual of photogrammetry* (Washington: American Society of Photogrammetry, 1952), 2nd ed., 876 p.

2. Schwidewsky, K. *An outline of photogrammetry*. Translated by J. Fosberry (London: Pitman, 1959), 326 p.

3. Heyll, H. Volume determinations in Rhenish open-pit lignite mines with the aid of aerial photogrammetry and electronic computing techniques. Translation by G. Richter. *Journal of the German Mine Surveyor's Association* (*Mitteilungen aus dem Markscheidewesen*), **67,** no. 4, 1960, 13 p.

622.1:526.918.52:622.013.36 *Paper 22*

Volume Assessment by Photogrammetric Methods

H. G. DAWE

Survey Production Manager, Hunting Surveys, Ltd., Boreham Wood, Hertfordshire, Great Britain

---

SYNOPSIS

Without going into the technical details of modern photogrammetry, the paper describes how numerical and/or graphical data for volume calculation may be derived from stereoscopic aerial photographs.

While it is not claimed that photogrammetry can in all cases match the accuracy and detail specification of the traditional surveyor working on the site, sensible comparisons can be made. It is recognized that measurements from air photographs, properly set up in suitable apparatus, meet all normal accuracy requirements for plans up to 40 ft to 1 in. scale for graphical presentation and contours may be plotted at intervals down to 1 ft.

The precision with which data may be read off a graphical plan or map is largely governed by the scale of that plan—the larger the scale the greater the precision—and at 1 to 500 scale the readability is in the order of ±3 to 6 in. Greater precision in the photographic and plotting processes is therefore rather academic and photogrammetric specifications are laid down for the final scale required.

When numerical data are more suitable for the engineer's task, however, the stereo observations may be recorded directly in the photogrammetric instrument and are not bound to the same extent by graphical scaling limitations. Modern instruments are equipped with electronic read-out in typed and punched-tape form for direct introduction to an electronic computer with a suitable program.

The precision of modern cameras and plotting instruments permits a discrete-point accuracy of ± 2 to 3 in. in height on clear ground which, coupled with the relative ease with which a very high density of points may be read, results in a high statistical accuracy being achieved—at least comparable with ground methods on most sites and superior to them in conditions of difficult, accidented terrain. No matter to what degree of precision field work is carried out, there is always, of necessity, a large proportion of the terrain that remains completely unobserved. Reducing its unknown extent provides a better statistical accuracy than merely increasing the precision on the same number of observations.

One of the most important factors in the aerial technique is the proper identification and interpretation of the mass to be measured and its freedom from obscuration. The subject must be clearly seen in order to be measured accurately and the photogrammetrist must be in possession of sufficient knowledge to enable him to differentiate between effective and non-effective areas. In certain cases supplementary information obtained on the site is necessary.

The position has not yet been reached where photogrammetric surveys can be carried out with high precision without a number of key control points being fixed by ground methods and meticulously identified on the relevant photographs.

Photogrammetric techniques are normal for a wide variety of engineering purposes today—ranging from reservoir capacity assessment to stockpile measurement and cut-and-fill on highways, etc., for record and design purposes—and only minor adaptation would seem to be necessary to widen the application to cover opencast mining and quarrying.

As in many other industries and professions, the introduction of electronic computing to aerial survey work in recent years has led to significant improvements in the methods used in the preparation of photogrammetric survey data—whether in the form of maps and plans, profiles and cross-sections or digital information on areas and volumetric capacities. The elimination of tedious manual computation has permitted the use of techniques which, by virtue of their mathematical content, were previously extremely time-consuming and uneconomic and could rarely be considered during the critical stages of engineering planning. Furthermore, so great is the speed of electronic computation that many alternatives can be calculated on the same basic data, thereby giving the engineer an entirely new approach and increasing his opportunity of making use of statistical data to an extent previously denied him.

The basic principles of deriving graphical and numerical data from aerial photographs have not materially changed and no instruments designed primarily for photogrammetric work have been rendered redundant or obsolete; on the contrary, they can be considered to have been given a new lease of life since their performance factors have been increased and their field of operation widened. The instruments in themselves are analogue computers and simulators into which it is possible to introduce stereoscopic air photographs in a manner that exactly reproduces the relative and absolute conditions of the camera in space. Thus a model of the terrain, on which readings may be made and recorded numerically or graphically at the will of the operator, is available. The speed with which such observations or plotting can be made is greatly in excess of ground methods and in any case such processes constitute only a relatively small proportion of the total work effort, which has already included the taking of the photographs, the provision of control points and the setting up in the instrument. Therefore, an increase in density of observations does not lead to a proportional increase in the cost of the survey, and may be only fractional. The advantage to the engineer of more detailed knowledge of a site is, however, very significant and the opportunity of obtaining a much larger number of values is a very real one.

The prime function of a photogrammetric plotting instrument is to simulate the flight conditions of the aerial camera at its moments of exposure, using pairs of adjacent overlapping photographs. The photographs themselves are used at their natural size (usually 9 in. by 9 in.), each component of the pair being carried in an identical camera unit which faithfully reproduces all the characteristics, or inner orientation of the actual air camera, at full size. The positions of these ‘dummy’ cameras in the instrument are so arranged that by optical or mechanical means the correct relationship can be established between the individual photographs with respect to the distance separating the exposure stations in the air and the altitude of the aeroplane. In other words, a full spatial reconstruction of the flight conditions is achieved and all distortions arising from tilts of the camera axis and differences in altitude are eliminated. The information necessary to set up the photographs in this way is not obtained from any recordings taken during the photographic flight, but is derived from parallax observations within the instrument and the known values of certain ground points identifiable on the photographs (Fig. 1).

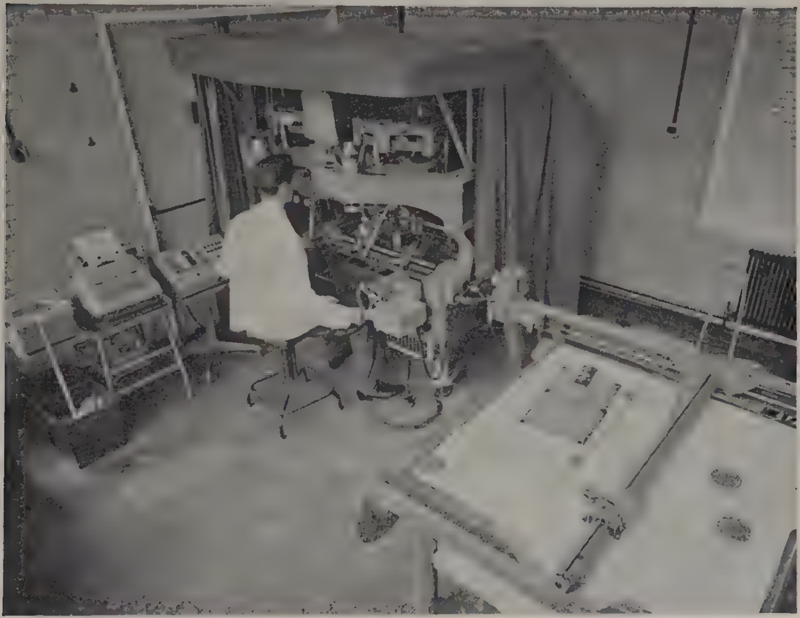


Fig. 1.—First-order plotting instrument showing facilities for recording data by typescript, punched tape or graphically by means of plotting table.

Once the photographs are properly set up it is possible to record in graphical or numerical form the salient facts concerning the particular piece of terrain covered. Distances may be measured and plotted at a suitable scale, absolute heights in relation to any given datum may be read off in the form of spot-heights or plotted as continuous contours and surface details added to complete the map or plan of the area concerned. During this process the third-dimensional photographic model is studied under high magnification by the operator, who controls his position and observation by means of a reference mark in the optical system which can be moved about in the horizontal plane and also adjusted vertically relative to the terrain model being viewed. The track followed by the reference mark is reproduced accurately by a pencil on a plotting table which is an integral part of the instrument.

The mechanical movements that take place on the table are a reflection of identical movements in the instrument, except that a scale ratio may be introduced between them.

It has always been possible, at any stage during the plotting of an overlap, to express the position of the reference mark in space, in terms of machine co-ordinates (X and Y horizontal and Z elevation) referred to an origin within the instrument. The numerical values defining the position of the reference mark, however, were normally only used for the setting and adjustment of the instrument prior to proceeding with a graphical plotting. The manual reading of three micrometer drums, each to six digits, and the desk calculations which followed rarely provided any information that the plotting pencil could not record rather better in much less time. Nowadays, however, the X, Y and Z co-ordinates are automatically recorded in typescript and punched tape simultaneously at the press of a button, electronic selectors having replaced the human visual system.

It is readily apparent that this development opens up many new possibilities in the determination of terrain data now no longer restricted by the limitations of purely graphical representation. The graphical plot or map has often been only the means to an end, particularly where profiles, cross-sections and volumetric matters are concerned and has usually involved a tedious 'taking-off' process, e.g. the measurement by planimeter of areas contained by successive contour levels or the interpolation of longitudinal and cross-sections. Since the processing of numerical data can be carried out rapidly and efficiently by electronic computer there is rarely need to rely on a contoured map or plan because the significant data can be supplied directly from automatically recorded observations. That the data so derived are at the same time more accurate is a further advantage.

To date, comparisons have been confined within the photogrammetric processes, but at this stage it is appropriate to consider how the matter stands in relation to conventional field survey. It is recognized that under certain conditions there is no substitute for the 'man on the ground' (the field surveyor), but it must also be acknowledged that developments in photogrammetry have enabled surveys to be carried out speedily and precisely in areas full of hazards or frustrations for the field surveyor. Photogrammetric processes have long been accepted as the better technique for topographical mapping. Between the clearly defined spheres where the superiority of one method over the other is unchallenged, there is a field of engineering survey in which ground methods or aerial techniques have been used, generally according to the conviction and experiences of the engineer involved. Timing and the non-availability of field staff have been the usual factors leading to a change from conventional methods to the 'newer' techniques but once made there is rarely a return to field methods and nowadays practically all engineering projects of any significance involve photogrammetry at some stage. Aerial photographs and their photogrammetric plotting figure more and more often from first-stage planning to completion.

Improved facilities for the rapid processing of data by electronic computer has not only extended the scope of photogrammetry within projects already committed to its use, but has led to its introduction to spheres where previously little or no advantage was to be gained from it. For many years photogrammetry was considered primarily as a tool for producing contour maps and plans for the engineer's use and a great deal of work so far as converting the graphical data into suitable form for statistical or design purposes remained. Today, however, data much nearer the end result come direct from the photogrammetric instrument/computer combination, having by-passed completely the preparation of the plan where it was only a means to an end.

As in all survey work, the characteristics of the area largely govern the method of attack and the type of information required and each case must therefore be treated on its merits. The true definition of a terrain surface is usually the main objective, whether it is in the form of a valley to be flooded to create a reservoir or a predetermined band within which a railway or a road is to be built.

Since data can be extracted in the form of profiles, contour lines or as spot-heights at predetermined intervals, the selection of the particular type of information to be used will depend on the size and shape of the area as well as on the height or depth of the capacities required.

The main uses to which photogrammetry and electronic computing are put at present are the volumetric assessment of coal stocks and reservoirs and the assessment of volumes of cut-and-fill, etc., connected with the construction of highways. There are, however, many similar uses to which the system can be applied, including opencast mining and quarrying.

On many quarrying and opencast mining projects a comprehensive plan showing all natural and artificial features, as well as contours and volumetric assessments, may be required. In such cases the air photography must be suitable for both planimetric and altimetric plotting to the accuracy required by the engineer.

Plans are usually required at 1 to 500 scale with contours at 1-ft vertical intervals, and in order to meet these specifications photography is taken from about 1250 ft giving a contact scale of 1 to 2500. Fig. 2,

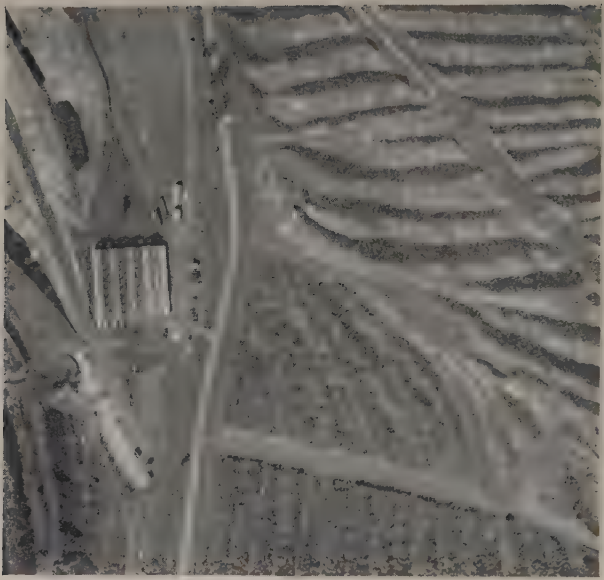


Fig. 3.—Aerial photograph of an opencast mining site.
(Also shown are the various stages of re-afforestation.)

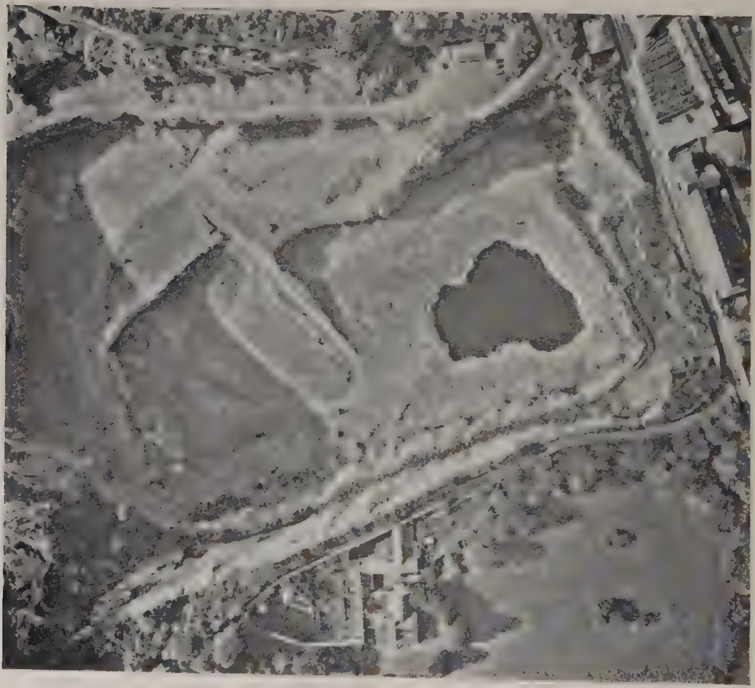


Fig. 2.—Part of a vertical aerial photograph of a quarry.

taken from about this height, illustrates the amount and complexity of detail which can be recorded under suitable conditions.

Where smaller-scale mapping is required the height from which photography is taken can be increased (Fig. 3).

Volumetric assessments can be made by the use of spot-heights read off at intervals in the form of a grid network, the horizontal interval between readings being governed mainly by the size and nature of the project. Very little computation is required since the whole project can be sub-divided into square columns, each column having a top surface area derived immediately from the grid interval and a height obtained from the spot-height level value. The volumes of the separate pillars are then integrated to give the volume of the whole mass.

Since the shape of the area will never conform to an exact grid network, clearly some additional readings must be taken around the limits of that area, but in practice it has been found that if one extra value is taken at every grid intersection outside the limits of the area an extremely precise result can be obtained, particularly bearing in mind the fact that near the outside limits of an area, whether it be a stockpile or reservoir, the volume contained within one single grid square is small compared with that within a similar size grid square nearer the centre.

A further refinement can be effected if, instead of accepting individual spot-height values as being the height of the column under consideration, adjacent spot-heights are taken into account and a mean value for the heights of the column is obtained (Fig. 4).

One particular problem connected with the computation of stockpile volumes is that of referring spot-height values to a particular datum. Inevitably stockpiling is carried out on uneven ground and unless an accurate shape of this ground can be obtained before stockpiling begins it is impossible to make a completely accurate assessment at a later date.

In taking off photogrammetric values readings are generally extended along both grid axes to include a sample portion of ground surrounding the stockpile and the ground configuration thus obtained is projected by extrapolation under the stockpile, thereby obtaining a reasonable ground surface shape. It must be remembered, however, that this is an extrapolation only and, although the upper surface of stockpiles can be measured precisely, it is not possible to be certain of accurate results when the ground surface is largely unknown.

Where stockpile volumes are required at frequent intervals for comparison purposes it is advantageous to establish ground markers with known values. These markers appear on the first and every subsequent set of photography so that all volumetric assessments for the stockpile under consideration can be referred to the same datum and any ambiguity which may arise from using different control values can be avoided.

With modern air-survey cameras and plotting equipment there is little or no difficulty in taking off measurements to the accuracy required and, since the computations involved are free from inaccuracies, the art of planning and executing volumetric assessment is concerned mainly with deciding on the scale of photography to be used and the frequency with which measurements are to be taken to ensure that a sufficiently accurate result is obtained with the minimum of time and cost.

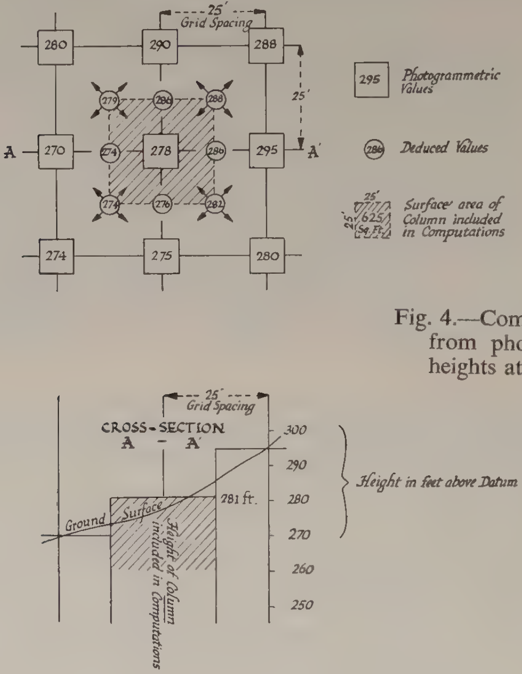


Fig. 4.—Computation of volumes from photogrammetric spot-heights at 25-ft grid intervals.

Where coal stocks are involved it is usually sufficient to take off height values of grid intersections at 25-ft horizontal spacing and any irregularities in surface shape too small to be accommodated at this interval are taken into account by the photogrammetric operator who assigns not necessarily the true value of the height but rather a representative value so that an overall surface configuration conforming more closely to the actual surface is depicted.

In the volumetric assessment of reservoirs a grid interval more compatible with the size of the area is selected—usually in conjunction with the engineer since he is able to state his accuracy requirements in such a way that the photogrammetrist can decide the frequency with which values must be taken in order to meet these requirements.

Where a topographic survey as well as volumetric assessment is required, it may be advantageous to produce contoured plans of the area and to compute the volumes from the information they give. The volumes may be computed by measurement of the areas contained within individual contour lines or by superimposing a grid and reading off spot-height values at grid intersections by interpolation from the contours. Both methods, however, involve a good deal of labour in addition to that required in producing the basic data and both are prone to random errors.

The introduction of the more sophisticated type of planimeter coupled with a read-out device or display may well encourage the use of contours in the assessment of areas and volumes, but at present the planimeter does not cover sufficient acreage from a single position to warrant its use over large areas.

Perhaps the most efficient and economic method of producing data for volume assessment is that of making a series of cross-sections at regular intervals over the whole area under consideration. Most first-order photogrammetric plotting instruments have the facility for transposing the Z or height movement with the X or northings movement and when this transposition has been effected the photogrammetric operator can trace out cross-sections directly on to the plotting table. It will be appreciated that if the reference mark is maintained in close and continuous contact with the surface of the stereoscopic model being measured the resultant cross-sections will include all minor changes in ground height and give an extremely detailed representation of the ground configuration. The system does not lend itself to electronic computation since the data are produced in graphical form and volumetric assessment must be made by first measuring the area of each and every cross-section by a planimeter or similar device.

As an alternative to the graphical representation of cross-sections, they can be defined by reading off height values at all changes of slope and when data are produced in this form they can be programmed for electronic computation to produce a volumetric assessment without further processing.

Experience has shown that the system of producing cross-sections in the form of three-dimensional co-ordinates is probably the most accurate and efficient for the assessment of volumetric capacities since it can be adapted to suit any size or shape of area and is completely independent of any errors inherent in graphical representation. Furthermore, the data produced by this system are eminently suitable for electronic computation and can be readily recorded in proper continuity for any particular computer program.

One particular advantage of the aerial survey techniques, not always fully appreciated, is that the quality of the information it supplies can be tailored precisely to fit the exact requirements for the particular project in hand. Although three-dimensional co-ordinate values can be obtained to within 3 to 6 in. in a first-order plotting instrument, their values can often be wasteful on projects not warranting such precise measurements and it is essential from the economic standpoint to ensure that the specifications governing the survey are realistic.

The most important single factor affecting the time, cost and accuracy of any survey or volumetric assessment is the flying height from which aerial photographs are taken. Photography at 1 to 2500 contact scale taken from approximately 1250 ft above mean ground level is suitable for the production of plans at 1 to 500 scale and contours at 1-ft vertical intervals. Height values from this scale of photography can be read to an accuracy of 3 to 6 in. and plan positions of readily identifiable points can be fixed to about the same order of accuracy.

If photography at a smaller scale is used, the accuracy of both height and plan measurements will decrease accordingly, for instance with photography at 1 to 50 000 scale taken from 25 000 ft above mean ground level, height values can only be read to about ± 5 ft.

Large-scale photography of a particular area will, of course, be more costly than that on the small scale since correspondingly more photographs will be required and more ground control and plotting instrument time will be needed to produce subsequent plans or volumetric assessments.

It is not possible to give accurate average costs for aerial survey and electronic computation since these vary considerably with the size and complexity of a particular survey, and each individual project must be treated separately.

A substantial part of the cost of obtaining aerial photography is in the positioning of the aircraft and the rate per photograph for small projects is correspondingly higher than that for larger projects. It is doubtful whether any project, however small, can be covered for less than £50. At the same time, if a number of small projects can be included in the same sortie the positioning cost for each separate project can be greatly reduced.

The supplying of ground control for photogrammetric plotting is carried out by normal survey methods and normal costs apply but, where possible, distance-measuring instruments and automatic levels are used in order to carry out this stage of work both quickly and economically.

The cost of photogrammetric plotting will depend on the size of the project, the amount of detail required, the contour interval and many other factors which vary with each project. The production of maps may cost from £4 to £40 per acre and the assessment of a stockpile would probably cost £150. Limitations to the scope and accuracy of photogrammetric values depend on the quality and scale of photographs as well as on the nature of the terrain.

In regions covered by vegetation photogrammetric observations are limited to areas which are visible from directly above and it is advisable to arrange photography during seasons where foliage is at its minimum in order to obtain the maximum results. The texture of the ground will often affect the accuracy of readings since, while it is possible to obtain height values to 3 in. on roads and similar surfaces, similar results cannot always be obtained on rough ground or pasture.

In order to obtain satisfactory photogrammetric results the photography used must always be of reasonably good quality. In the United Kingdom weather conditions impose a good deal of restriction on photographic flying, but in general photography is possible in good weather throughout the year. Obviously, the summer period is by far the most suitable, but where foliage or vegetation cause difficulties in plotting it is sometimes advantageous to photograph during the winter or early spring. Plotting from winter photography is frequently hampered by the presence of long and deep shadows and it is obviously dangerous to attempt plotting in areas covered by snow, but quite often a ground frost can produce lighting conditions particularly suitable for photography.

For large-scale surveys photography is required from a height below the cloud base generally prevailing and with high-speed aerial film and modern cameras with fast shutter speeds suitable photography can often be obtained without cloud-free skies.

All first-order plotting instruments can accommodate the normal tilts invariably occurring with vertical air photography and a certain degree of air turbulence can therefore be tolerated in survey flying. Rough weather conditions can, however, prevent photography being undertaken.

As far as is known, there are no legal objections to obtaining air photography. There are, however, areas where photography is not permitted and all survey aircraft must conform to the normal flying regulations relating to civil and military air corridors, etc. It is usual for the company operating the aircraft to hold the copyright of the photography, but in its own interests and those of its clients it would not divulge information or publish results without permission.

The use of electronic printing devices has greatly improved the quality of contact prints and diapositives in that highlight shadow details are brought out more clearly but, nevertheless, it is not always possible to obtain first-rate photography and speed and economy sometimes demand that surveys have to be carried out under other than optimum conditions.

Considerable research covering all aspects of photogrammetry and electronic computation is at present being undertaken and undoubtedly new and more efficient techniques will emerge in the not too-distant future. The use of infra-red and colour photography is under consideration with a view to obtaining more qualitative as well as quantitative information from aerial photographs and, although it is doubtful if such processes will give more accurate results, the scope of volumetric assessment may well be widened.

A study of the type and amount of work at present in progress on volumetric assessment shows that photogrammetry and electronic computation are already making a valuable contribution in this field and, provided current interest is maintained by civil engineers and photogrammetrists, this contribution will increase in the future.

621.3:622.271.3 *Paper 23*

Electrical Planning for Large Opencast Mines

T. ATKINSON, M.I.E.E., M.I.Min.E., A.M.I.Mech.E.

Senior Engineer, Powell Duffryn Technical Services, Ltd., Consulting Engineers

---

SYNOPSIS

Over the past decade opencast mining machinery has tended to increase in size to enable deeper deposits to be mined economically at high outputs. This, and the fact that more belt conveyors and large trolley-wire locomotives are being used, make even more necessary carefully planned electrical distribution systems.

In many opencast mines the electrical distribution system must be designed for progressive advancement as the mine face advances, and the conflicting needs of flexibility, safety, capacity, transportability and reliability must be resolved. Individual machine loads of up to 20 000 kVA and pit distribution voltage of 35 000 V requiring connection to electrical utility grid supplies must be catered for. Many mines have electrical loads equal to that of towns of medium size.

The paper discusses the detailed planning of surface transmission, pit electrical layout, system design, distribution equipment design, trailing cable selection and handling, electrical protection, mining machinery electrical equipment, plant commissioning, emergency supplies, lighting, and ancillary supplies. The special requirements imposed by mining methods, climatic conditions, national electrical regulations, flooding and groundwater hazards, maintenance, and staff training, are included.

The equipment and techniques that have been developed in Great Britain, America and on the Continent to provide flexibility, reliability, safety and transportability are mentioned.

Over the past fifteen years opencast mining machinery has shown a marked tendency to increase in size to enable deeper deposits to be mined economically at high outputs. Generally the use of the electrical drive for these large machines has brought substantial operating economies, but has presented certain problems in power supply and distribution which must be solved at the planning stage. The conflicting needs of capacity, flexibility, transportability, reliability and safety must be resolved.

The prime function of the electrical distribution system is to provide an electricity supply of adequate capacity, at the required voltage and frequency, at the supply points. Many opencast mines are characterized by rapidly advancing faces. The electrical distribution system must be then planned so as to be capable of progressive advancement as the mine face advances. With the increasing use of high-speed side-slewable belt conveyors and trolley-wire locomotive haulage to handle the outputs of large-scale excavating machinery,[1] the need for carefully planned electrical distribution systems has assumed an increasing importance.

POWER SUPPLIES

In general, the power supply to a large opencast mine must be of a very high order of security for the following reasons:

(1) the magnitude of the project may be such that loss of production is very expensive;
(2) mining operations are often continuous and therefore lost time cannot be made up; and
(3) the security of the mine may rely upon vital services, such as continuous pumping.

For the best possible security without excessive capital expenditure the following are usually provided:

(i) duplicate feeders from the source of power to the mine, if possible by separate routes;
(ii) ring-main system in the mine with sectionalizing ring-main circuit-breakers: this must, of course, depend to some extent on the layout and method of operation of the mine; and
(iii) local standby supplies for vital services.

Where overhead feeders are employed, hazards such as lightning or hurricanes must be taken into account in the design of the line. The stringing of shielding, overhead earth wires, the application of surge diverters and rod gaps, and the routing of the line to avoid exposure should be carried out. Where the feeders are the property of the electricity supply utility, the authority should be consulted so that the best possible protection is obtained.

Many large mines receive their power supplies from the local electricity supply utility or, in the case of some large producers of fuel, from an associated thermal power station which is usually interconnected with some form of electricity supply network. In general, these stations follow central station practice and are fully described elsewhere.[2]

In some locations, however, owing to the remoteness of the operation or the comparatively small size of the local utility, it may be necessary to provide a generating station to supply the mine. The selection of the type of prime mover depends upon the availability of fuel, cooling water, site conditions, etc. The choice of station is adequately covered in the literature.[3]

In certain circumstances, due to the mining conditions, the source of power must be moved from time to time and here the high-speed diesel engine using distillate-type fuel is usually employed. Where hazards

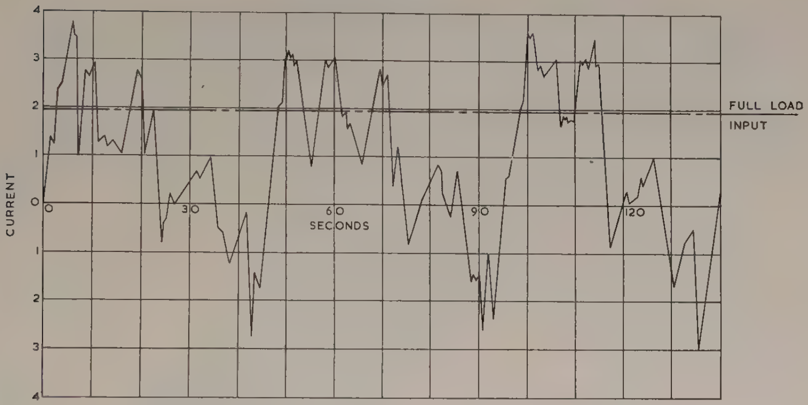


Fig. 1.—Current input to a typical shovel excavator.

due to flooding, etc., may occur, this form of prime mover has certain advantages for standby power supplies; these are described later in the paper.

In the early stages of planning, the effect of the mine load on the electricity supply network must be considered since it may have some undesirable characteristics. The electricity supply authority must be consulted. The operation of large draglines, shovels, etc.—i.e. machines of cyclic operation—may result in considerable load peaks and periods of regeneration when power is fed back into the system. Bucket-wheel excavators, bucket-chain excavators, and their associated equipment—i.e. continuously operating machines—do not generally result in such

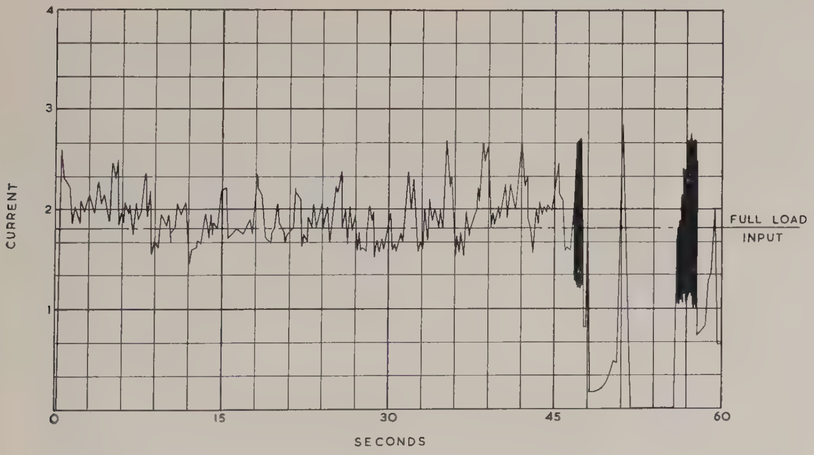


Fig. 2.—Typical current record for bucket-wheel excavator.

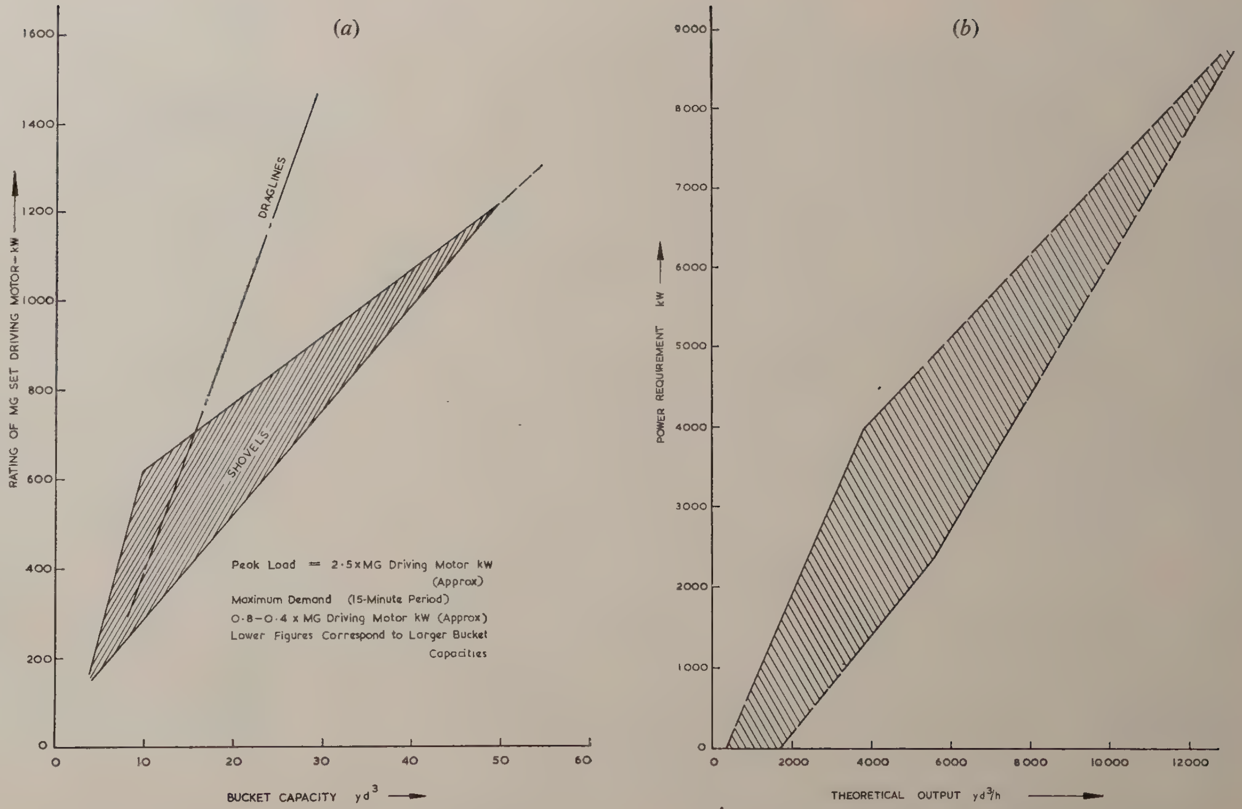


Fig. 3.—(*a*) Power requirements for excavators. (*b*) Power requirements for bucket-wheel excavators.

marked variation as draglines and shovels, but their performance must still be closely investigated. Figs. 1 and 2 show typical load cycles for shovel and bucket-wheel excavators. An approximation of the electrical load for the mine should be prepared using suitable diversity factors. Figs. 3(*a*) and (*b*) indicate the approximate power requirements for excavating machinery. Fig. 4 shows the variation in diversity for machines connected to the same source of supply. Some discretion is necessary in applying diversity factors, and each case should be assessed on the mode of operation to be employed.

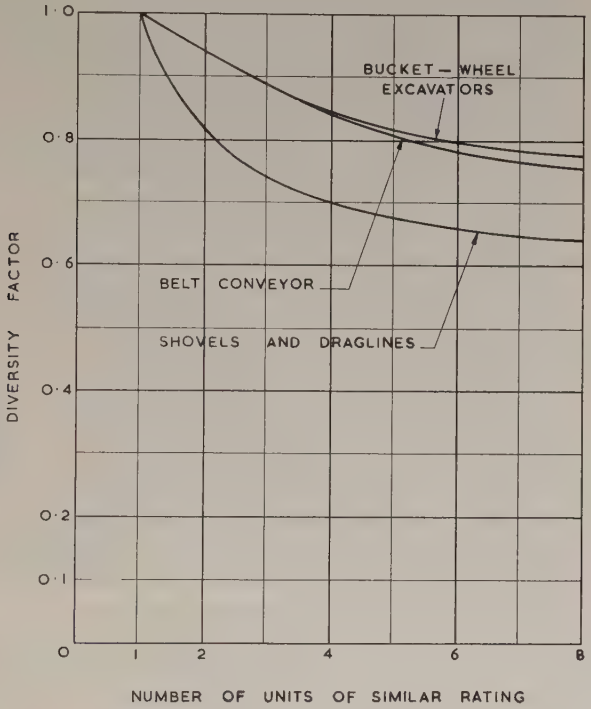


Fig. 4.—Diversity factors for units connected to same power system.

Good voltage regulation is of the utmost importance in order to ensure trouble-free and efficient performance of the electrical system. Low voltage can result in overheating of electrical machines and it becomes necessary to run them at reduced output. Invariably some form of compensation is required, this usually being the provision of on-load tap changers in the supply transformers. Where transformers operate in parallel, tap changers should be of the 'non-hunting' type.

Where the electricity network supplying the mine is relatively small, voltage variation and instability may occur. To compensate for this the electric utility may install series capacitors in the incoming lines, which have the effect of continuously regulating the voltage.[4] Where this type of compensation is used, it should be ensured that the reduction in circuit inductive reactance due to the series capacitors is not sufficient to cause 'hunting' of synchronous machines. The ratio of resistance to reactance $\frac{R}{X}$ in most feeders is low, however, and some degree of correction is, therefore, usually possible. The electrical machinery manufacturers should be consulted on this point.

The economics of maintaining the power factor of the mine load close to unity are well known. Where large motor-generator sets are employed on excavating machinery, synchronous-type motors are usually used with automatic control of the excitation to provide a constant power factor of unity or, in some cases, over-compensation is employed to improve voltage regulation. Conveyors, etc., are, however, usually driven by induction motors, having power factors less than unity. These machines may be compensated by connecting power factor correction capacitors across their terminals; or the load as a whole may be compensated by the installation of an automatically controlled high-voltage capacitor bank at the supply point. The economics of both methods should be investigated. Where a number of relatively large induction motors are employed it is usually not advisable to fit individual power factor correction capacitors, as modern machines may operate at power factors greater than 0·9, so that only small individual compensations can be effected if mechanical damage, due to self-excitation when a high inertia drive is switched off, is to be avoided.

In calculating operating costs, the overall cost per kWh of electricity must be established. Maximum demand, load factor, etc., must be taken into account and where an outside supply is being used, consultation with the electric utility is essential. Excavating machinery manufacturers usually provide figures for the electricity consumption of their equipment. Conveyors and trolley-wire locomotive installations should be assessed on volume of material moved rather than on installed capacity.

Table I gives typical annual depreciation figures for electrical equipment.

TABLE I.—*Typical annual depreciation figures for electrical plant*

	per cent		*per cent*
Power transformers	5	Trailing cables (drum-wound)	20
Circuit breakers	5	Trolley-wire locomotives	5
Distribution transformers	7	Trolley-wire rectifier equipment	7
Kiosk-type equipment	8	Trolley-wire O/H equipment	10
O/H lines	5	Lighting equipment	12
Flexible distribution cables	15	Communications equipment	8

MAIN INCOMING SUBSTATIONS

The location of the main incoming substation must be determined by taking into account the centre of gravity of the mine load and its future movement, physical limitations due to incoming feeders, hazards due to movement of large mining equipment, lightning, hurricanes, blasting, the sterilization of the area occupied, etc. In many instances it is possible to locate the main incoming substation permanently throughout the life of the mine.

Where mining operations necessitate the movement of the main incoming substation at a future date, careful planning is necessary to avoid disruption of supply. The opportunity may also be taken to accommodate future increases in supply capacity. On many occasions it may be economic to install additional equipment at a new location sooner than is required, so that the mine load can be gradually transferred without serious disruption. On completion of the change-over, the equipment from the first substation can be moved to the second station in anticipation of the proposed load increase or to the next location in the case of rapid advance.

The electrical demand of most large mines is such that the equipment used in the main incoming substation is not readily transportable. Electric utility practice is usually applied with outdoor-type gear predominating. One development which is now finding favour is the 'packaged-type' substation using indoor switchgear in weatherproof kiosks. This equipment has certain advantages in reduced erection time and civil works. The equipment is well established for use in 33-kV systems and is available at 66 kV.

PRIMARY DISTRIBUTION

The method of primary distribution from the main to the distribution substations is dependent upon the magnitude of the load, the mining method and the degree of security required. Where large loads must be distributed over long distances it is usually more economic to use overhead lines than insulated cables. Overhead lines are considerably more vulnerable to lightning than cables and where the isoceraunic level is relatively high it is necessary to protect the system by fitting shielding, overhead wires, surge diverters and rod gaps. Co-ordination of insulation levels throughout the system must be given due attention as obviously that of the apparatus must be greater than the breakdown level of the protective devices.[5] The basic insulation level of all the equipment must be ascertained, surge diverters specified to valve off at a suitable value, and the settings of the rod gaps determined. Shielding overhead earth wires should be installed above the overhead lines, earthed at every pole and connected to the substation earth networks. As a guide to obtain adequate shielding, the earth wire should be positioned so that a line drawn through it and the farthest power conductor does not subtend more than 30° to the vertical.

Insulated cables must be used for relatively short distances, or where the movement of large mining machinery from the erection yard to the mine and physical obstructions eliminate the use of overhead lines. Owing to the need for flexibility in cable handling, the types of cable used by electric utility undertakings are generally unsuitable. The cable must be readily recoverable, with the result that the use of armouring presents difficulties. As the cable is usually outside the mine excavation, provided the route of the cable is well designated the armouring

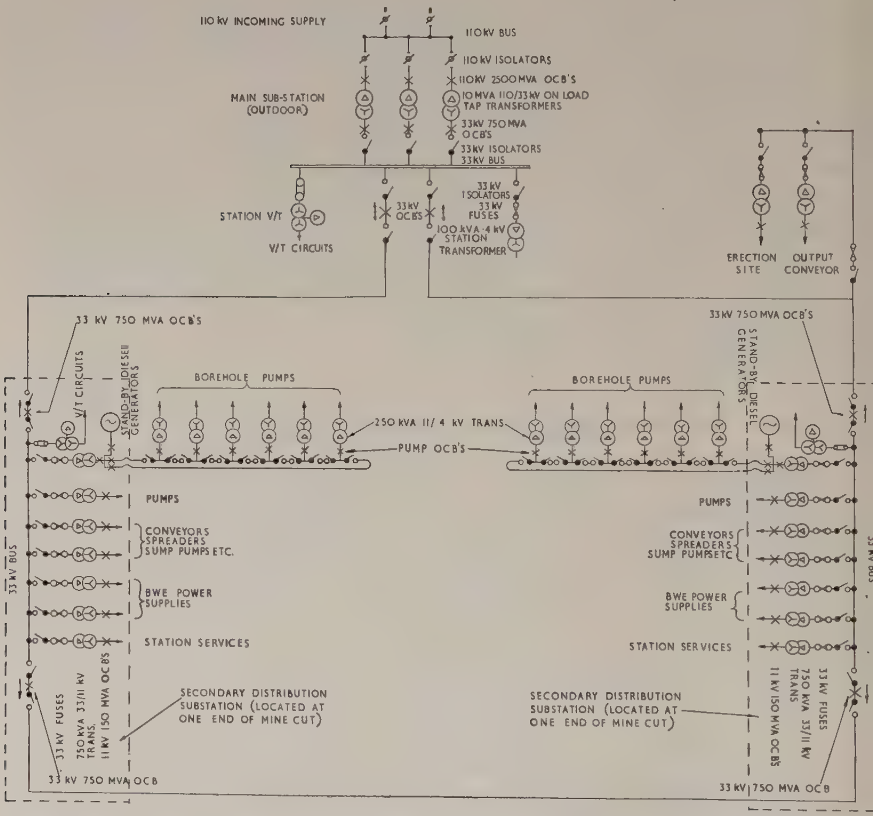


Fig. 5.—Typical schematic electrical distribution for adopting the 'leap-frogging' method.

can be eliminated, subject, of course, to compliance with national electrical regulations. Single-core polythene-insulated cables are in use in the U.S.A. for voltages up to 33 kV (to 35 kV in Germany). Special techniques are necessary for the jointing and termination of these cables, and they should be purchased fitted with terminal equipment where possible. Where cables must be terminated on site, it is advantageous to train a member of the mine staff in the necessary techniques during the commissioning period. The degree of skill required is probably less than that required for paper-insulated cables.

The selection of the primary distribution voltage is of considerable importance. The economics of each case must be considered, due consideration being given to voltage regulation, insulation problems, need for flexibility, etc. Primary distribution voltages of 33 kV are common and 66 kV may have to be adopted for very large loads.

It may be necessary to install several distribution substations and, where possible, one or more ring-main systems may be employed with advantage. The ring-main system has the advantage of greater security of supply, increased flexibility in operation (particularly where periodic advances of the substations must be carried out) and better voltage regulation. The ring main may be sectionalized by the use of ring-main circuit-breakers equipped with directional-type protective devices so that a faulty section of the ring main may be isolated without interfering with the supply to the remainder of the system (Fig. 5).

The location of the distribution substations requires considerable investigation in order to provide the maximum flexibility of electricity supply without the need for re-location at too close intervals of time. Where a 'swinging-face' pit is being operated, it is obviously an advantage to locate the distribution substation at the swing point. If 'parallel advance' is the method of working, substations should be installed at each end of the pit, well ahead of the face. The face eventually advances well beyond the substation and the substation must then be re-located ahead of the face.

Where parallel advance working is being carried out and security of supply is of paramount importance, a system of 'leap-frogging' can be adopted. Fig. 5 shows a typical layout of an electrical installation using this system. A 33-kV overhead line forms a ring main around the mine. Two distribution substations are located, one at each end of the mine. Ring-main circuit-breakers of the outdoor type are located at both ends of the substations and connect to a 33-kV overhead busbar. The 1750-kVA feeder transformers are supplied from the overhead busbar through ganged, air-break, disconnect switches and 33-kV liquid fuses to eliminate the high cost of circuit-breakers. The transformer size is selected to cover the largest single item of equipment, and other loads are grouped to provide a reasonable proportion of

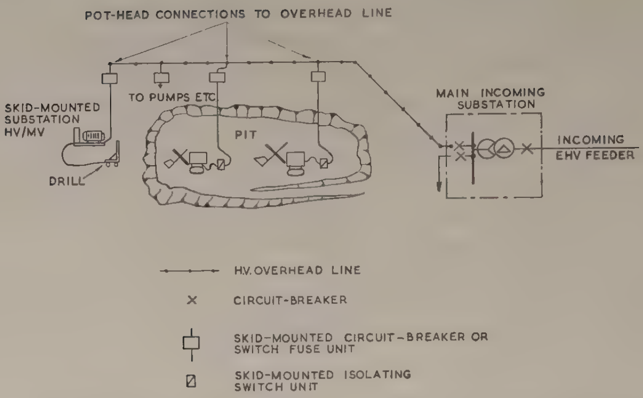

POT-HEAD CONNECTIONS TO OVERHEAD LINE
SKID-MOUNTED SUBSTATION HV/MV
TO PUMPS ETC
PIT
DRILL
MAIN INCOMING SUBSTATION
INCOMING EHV FEEDER
HV. OVERHEAD LINE
CIRCUIT-BREAKER
SKID-MOUNTED CIRCUIT-BREAKER OR SWITCH FUSE UNIT
SKID-MOUNTED ISOLATING SWITCH UNIT

Fig. 6.—Overhead line distribution system where security of supply is not vital.

the transformer capacity. The transformers are not arranged for parallel operation, in order (*a*) to simplify movement of the substation, (*b*) to restrict the prospective fault level, and (*c*) in this instance to eliminate the need for neutral earthing switches and resistors.

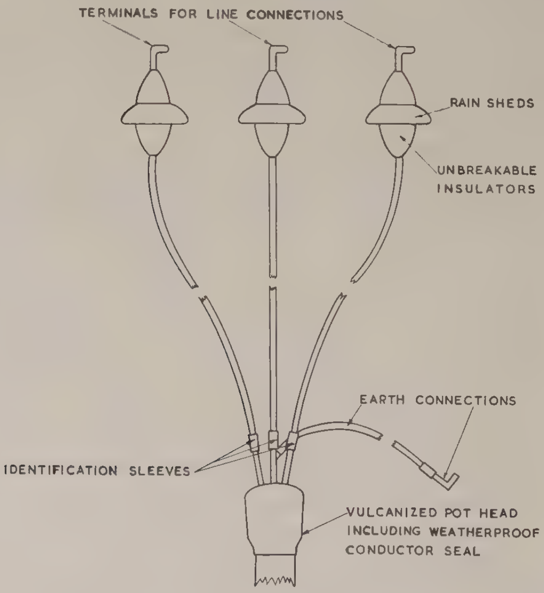

TERMINALS FOR LINE CONNECTIONS
RAIN SHEDS
UNBREAKABLE INSULATORS
EARTH CONNECTIONS
IDENTIFICATION SLEEVES
VULCANIZED POT HEAD INCLUDING WEATHERPROOF CONDUCTOR SEAL

Fig. 7.—Vulcanized pot head for connecting trailing cable to overhead lines.

The restriction of fault level enables the pit distribution cable size to be determined on current-carrying capacity and voltage-drop requirements, and not short-circuit rating, resulting in the use of a less expensive cable of smaller diameter which is more easily handled. The transformers are fitted with adaptors on the 11 kV side to accept the plug couplers of the 11-kV circuit-breaker cables. A spare transformer and 11-kV circuit-breaker are provided so that, in the event of a failure, the affected feeder cable can be transferred quickly. When it is necessary to advance the substation, this may be done, one transformer at a time, without undue disruption of supply.

Where security of supply is not vitally important, such as an operation where a number of machines are working independently and a brief shutdown of one would not seriously jeopardize the output of the mine, the distribution system may take the form of a single overhead line erected close to the mining area (Fig. 6). Machines are then fed by trailing cables, through a circuit-breaker, or switch fuse, connected to the overhead line by a short length of trailing cable terminated with a vulcanized pot head (Fig. 7). Each tail of the pot head can, if necessary, be fitted with a 'live-line' clamp that can be connected to the overhead line by an insulated operating stick without making the line dead.

PIT DISTRIBUTION

The choice of *pit distribution voltage* is one of economics, due regard being given to the type of equipment available. The tendency is towards higher pit distribution voltages to enable power to be transmitted economically to the large mining units now being installed. In Europe pit distribution voltages of 15 kV and 25 kV are commonly used in the brown coal mines, and 35 kV has been adopted in some cases where very large excavators and overburden bridges are in use. In the British Commonwealth voltages of 11 kV and 13·2 kV are in use, while 7·2 kV is in common use in the U.S.A., but the tendency is to higher voltages as machine sizes increase.

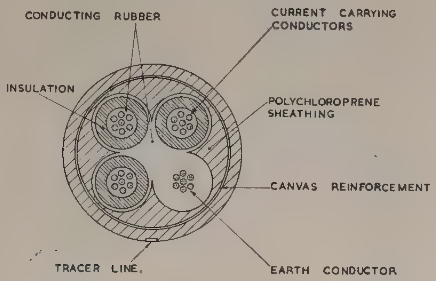
The choice is mainly an economic one, the operating costs being the main criterion. As a general rule higher voltage equipment becomes more favourable the higher the kVA loading.

There has been considerable controversy about the method of earthing the star points of the pit distribution transformers. Often the electric utility undertaking's practice is followed and in most countries there are government requirements to fulfil. Careful consideration must be given to the problem since some of the conditions arising from the use of mobile mining machines do not occur in electric utility practice. The most important consideration is the contact resistance between a machine and earth. Where a machine travels on a hard dry rock surface, particularly where the rock is of high electrical resistivity, the voltage-contact factor between the machine and earth can be such that a person standing beside the machine and touching it can receive a lethal electric shock if an earth fault occurs, even though the machine is properly earthed. The low bearing pressures characteristic of well designed modern machines tend to worsen this condition. The methods of approach to these problems are fully discussed elsewhere.[6,7,8] The usual procedure is to insert some form of impedance between the transformer secondary star point and earth to restrict the earth fault current. The value of this impedance, however, should not limit the fault current to such a value that the operation of protective relays is made difficult. The continuous monitoring of earth loops can be carried out by the use of one of the earth cores in the trailing cables as a pilot wire, and this practice has been adopted to some extent in the U.S.A.

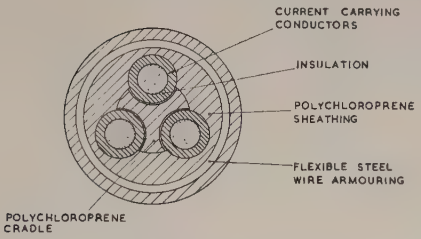
Where it is decided to earth the neutral point solidly, the earthing arrangements in the secondary distribution substations should follow electric utility practice. Grading wires should be buried around the periphery of the substation, and an earth network in the form of buried wires or a wire mesh installed. These should be solidly connected to the station earth and all metal work. All switch operation positions should be provided with an earth mat connected by a flexible conductor to the operating handle. Attempts have been made to provide additional earths for mobile machinery by driving temporary earth rods and connecting these to the machine, or by the laying of wire-mesh mats around the machine, but these measures have not been found to be effective or practicable.

From *secondary distribution substations* to the load points in the pit, flexible cables form the distribution link. These cables must have the following mechanical properties: (*a*) physical strength and resistance to abrasion; (*b*) flexibility; and (*c*) resistance to the environmental and chemical conditions experienced at the mine.

In Great Britain (and to a lesser extent in the U.S.A.) cables of the 'pliable armoured' type, having a number of wires forming each strand of the armouring, are used. They are capable of withstanding comparatively rough handling. The tendency is, however, towards the use of unarmoured, individually screened, trailing cables. The screening may be a metallic braid or a conducting rubber compound. Cable insulation above about 7 kV is usually a butyl rubber or similar material to withstand ozone deterioration. To reduce corona discharge the conductor diameter may be made larger and more uniform by the application of a layer of conducting tape or extruded conducting rubber. The outer surface of the conductor insulation is treated similarly, resulting in a radial electric field of lower intensity around the conductor, this being less conducive to corona discharge. The overall sheath may be of canvas-reinforced, polychloroprene or nitrile rubber for installations where particularly high temperatures are experienced. Typical cross-sections of cables are shown in Fig. 8.



TYPICAL H.V UNARMOURED TRAILING CABLE


CURRENT CARRYING CONDUCTORS
INSULATION
POLYCHLOROPRENE SHEATHING
FLEXIBLE STEEL WIRE ARMOURING
POLYCHLOROPRENE CRADLE

TYPICAL PLIABLE ARMOURED CABLE

Fig. 8.—Typical cable cross-sections.

In calculating the conductor cross-section the following items must be taken into account: (1) circuit loading; (2) voltage drop; (3) short-circuit rating; and (4) derating due to high ambient temperature. Where relatively small transformers are used, such as in the 'leap-frog' system previously described, the conductor cross-section may not be influenced by the short-circuit rating, and economy in copper may be achieved. Voltage drop calculations should be carried out for the cable, but the system must, of course, be considered as a whole. In the tropics, where cables are exposed to direct sunlight, the black-bulb temperature may exceed 140°F and the cables should be suitably derated for this duty, as well as being made resistant to strong sunlight.

Consultation with the cable manufacturers should take place during the planning stage. For easy identification on mine benches, cables should have a helical yellow strip vulcanized along their length.

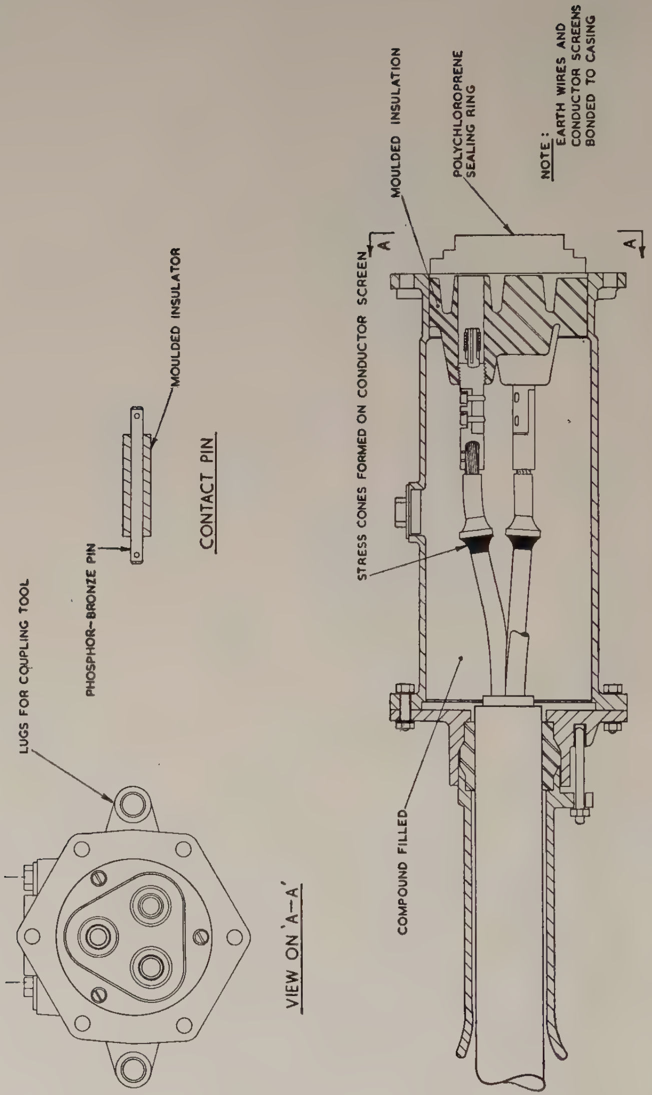

LUGS FOR COUPLING TOOL
PHOSPHOR-BRONZE PIN
MOULDED INSULATOR
CONTACT PIN
VIEW ON 'A-A'
STRESS CONES FORMED ON CONDUCTOR SCREEN
A
MOULDED INSULATION
POLYCHLOROPRENE SEALING RING
NOTE: EARTH WIRES AND CONDUCTOR SCREENS BONDED TO CASING
A
COMPOUND FILLED

Fig. 9.—11-kV 100-A half coupler of contact pin and tube type.

The jointing of cables is of particular importance. There are three main methods in use: (*a*) cable junction boxes; (*b*) vulcanized joints; and (*c*) plug couplers of the contact pin and tube type.

Cable junction boxes have been in use over a long period and have proved reliable, but they do not lend themselves to flexibility in operation and at least part of the work of making a joint must be carried out in the field. Vulcanized joints of various types and cable-capping methods are in common use in Europe and the U.S.A., but also do not lend themselves to flexibility and a large part of making a joint must be carried out in the field. Plug couplers are in use throughout the world for voltages up to 35 kV. Although more expensive in first cost, they can be used repeatedly and are particularly suitable for a flexible electrical system. The major part of the jointing work can be carried out by the manufacturers or in the mine workshops, and the plug couplers may then be jointed and broken in the field by a simple mechanical operation as often as necessary. Where the mine face is continually advancing, their use becomes very desirable. For maximum flexibility and interchangeability standardization is essential, and there is considerable advantage in selecting one cable size and one plug coupler for the whole of the pit distribution system. Fig. 9 shows a typical 11-kV plug half coupler.

Electrical distribution equipment may then be fitted with adaptors for receiving the plug couplers. Where the plug couplers of the contact pin and tube type are used, correct polarity of the adaptor terminals is essential to ensure that phases cannot become crossed in ring-main circuits. Fig. 10 shows a typical arrangement. When connecting the cable to the plug coupler the blue conductor is fixed to the terminal at the top. The other two cores are connected as laid up in the cable, no attempt being made to cross the cores inside the plug coupler body. This ensures correct polarity.

The cable layout in the pit is largely dependent upon the mining method, hazards such as rock falls being taken into account. Where possible cables should be laid parallel with the bench, and when crossing a bench should be raised on supports to avoid machinery passing over them. For vehicular access, transportable two-section cable ramps or portal-type supports should be provided. It is desirable, but not always possible, to avoid stringing a cable down a working face, and where practicable the cables should enter the pit over the slopes at the end of the pit.

Where a machine traverses a face it is usually convenient to provide a supply point at the centre of the face, most machines being provided with motorized trailing-cable drums, possibly built into the machine, and accommodating cable lengths of up to 3000 ft. If this length is exceeded problems arise owing to excessive weight and the need to derate the cable because of the number of layers on the drum, when separate cable drums become necessary. A typical bench layout is shown in Fig. 11. Where very long faces are planned there can be advantages and economies in the provision of two self-propelled cable drums of relatively small size operating with one machine.[9]

The initial handling of cables, particularly when 'walking' excavators from the erection yard into the pit, should be planned and supervised by an electrical engineer. Care at this stage can result in a considerable extension of cable life. Cable-drum transporters greatly reduce the manual effort of cable laying and recovery and in some mines are also used for conveyor-belt handling. Where labour is inexpensive cables are still manhandled, but close supervision is required and it is necessary to train gangs during the development stages so that they learn to appreciate the physical limitations of cables. Where they are laid on slopes, the cables should be lowered from a drum, sheaves being provided to protect the cable where abrasion may occur. There is often

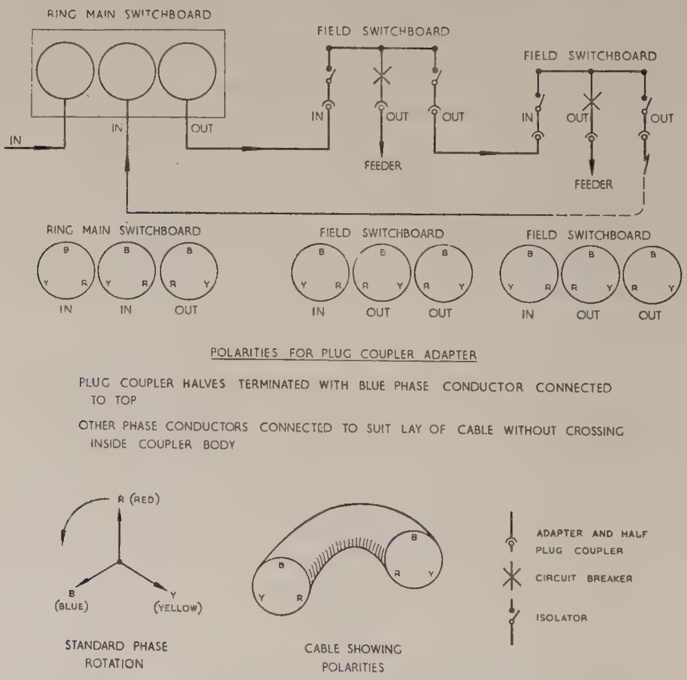
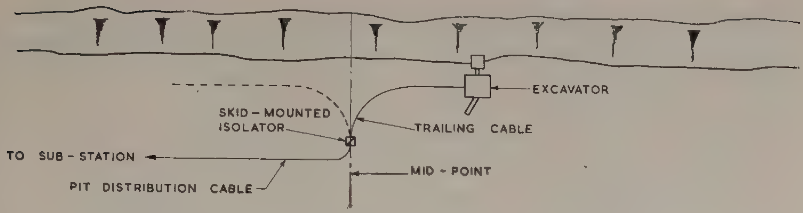
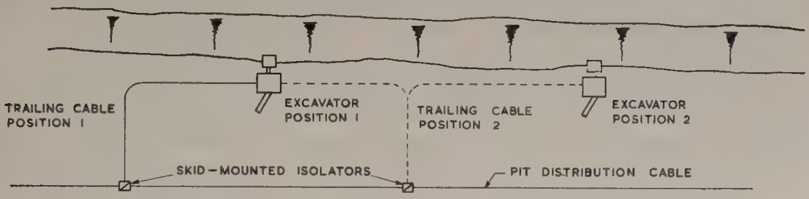


Fig. 10.—Polarity diagram for plug couplers and adaptors.



(*a*) Bench layout using supply point at mid-point of bench.



(*b*) Bench layout using 2 mobile reeling carriages.

Fig. 11.—Typical bench layouts using cable-reeling drums.

advantage in laying the cable across the slope to avoid the full dip; some support is usually necessary. Where equipment such as a bench-lift conveyor is installed, cables may be strung on its structure, plug couplers being provided at upper and lower bench levels. Cables should never be dragged by machinery.

SELECTION OF DISTRIBUTION EQUIPMENT

Distribution and control equipment for opencast mining operations must meet the differing requirements of transportability, robustness, flexibility of operation, safety, etc., and equipment must operate in all types of weather conditions. Dust is often a problem.

Skid-mounted kiosk-type units are in widespread use. The kiosks are weatherproof and form an additional dust protection barrier for the equipment. One manufacturer produces an outdoor oil circuit-breaker unit which is truly weatherproof and, as its centre of gravity is near to the ground, it makes an excellent opencast mining unit. Most units are purpose-built using standard items of equipment. In the U.S.A. 'transportable-type' sub-stations are commonly used,[7,8] while in Europe and the Commonwealth kiosk-enclosed equipment is in general use both for distribution and control equipment. Where conveyor-control gear is required the kiosk usually houses the high-voltage switchgear and the conveyor-control gear, an outdoor-type transformer usually being mounted on the same skids. To achieve maximum flexibility, both distribution and control equipment fed at pit distribution voltage should be fitted with 'in' and 'out' isolating switches (as shown in Fig. 10), usually arranged with an earthing position and fitted with neon indicators to enable operators to check that the line is dead before closing to earth.

It is usual in control equipment for auxiliary supplies to be provided for lighting, portable tools, control circuits, etc.

Circuit-breakers fitted to equipment used in the Commonwealth are usually of the bulk oil type with compound-filled busbar chambers. In the U.S.A. oil circuit-breakers are in general use, but in many cases high-voltage switch fuses are used. On the Continent expulsion-type and small-volume oil circuit-breakers are usually selected.

The requirements for switchgear for opencast mining are (*a*) robustness, (*b*) lightness for transportability, (*c*) low centre of gravity and (*d*) ability to operate without excessive maintenance. It is difficult to satisfy both (*a*) and (*b*). Lightness has usually to be sacrificed to some extent to obtain a practical mining unit. A low centre of gravity results in a more stable switchgear unit and may result in small, lighter skids. Air-break switchgear is generally lighter than comparable oil-filled units, but oil circuit-breakers are usually more suitable for dusty situations, the air spaces, and consequently 'breathing', being less so reducing maintenance. Air-break units, however, will probably be developed in the future. There have been prolonged arguments concerning the relative merits of air- or compound-filled busbar chambers: while air-filled units are lighter, for dusty humid situations there is advantage in selecting compound-filled units.

Industrial-type transformers are in general use and satisfactory service may be expected provided due attention is given to high ambient temperatures. Cold-rolled grain-orientated steels for transformer cores have now been proved in service and result in some weight saving. In the U.S.A. special transformers of reduced impedance, giving better voltage regulation, are obtainable, but their effect of increasing prospective fault level must be considered. Dry-type nitrogen-filled transformers have been given limited application and have been successful for voltages up to 11 kV. Because of their reduced weight there will undoubtedly be further developments with this unit as more knowledge is gained.

Control gear employing industrial-type contactors, etc., gives satisfactory service provided ambient temperature, dust protection, and so on, are allowed for.

Standard heavy-duty weatherproof plug-coupler sockets are generally suitable for medium- and low-voltage outlets. Again there is considerable advantage in standardization.

Table II gives approximate prices of the main electrical equipment needed for the modern opencast mine.

TABLE II.—*Approximate prices (1963) of opencast mining electrical equipment*

	£ Sterling
3·3-kV Circuit-breaker 1200A 150 MVA	950
6·6-kV Circuit-breaker 800A 150 MVA	600
11-kV Circuit-breaker 400A 150 MVA	520
3·3/·4-kV 1000 kVA Transformer	1200
6·6/·4-kV, 1000 kVA Transformer	1200
11/·4-kV 1000 kVA Transformer	1200
3·3-kV Flexible distribution cables 264A	4000 per 1000 yards
6·6-kV Flexible distribution cables 132A	3700 per 1000 yards
11-kV Flexible distribution cables 79A	3200 per 1000 yards

PROTECTIVE DEVICES

The protection of equipment and operators in the event of an electrical system fault requires careful planning, the system being co-ordinated as a whole and all protective relays graded back to the main supply point.[10]

In many installations a variety of different protective devices may be used and discrimination may be difficult. Ring-main circuit-breakers are usually protected by directional overcurrent and earth fault relays of the induction type with the tripping operation initiated by a battery. It is generally not difficult to grade induction-type relays, current-operated relays with shunt fuses, and liquid fuses, but it is difficult to grade magnetic relays fitted with dashpots with the devices previously mentioned, although some improvement in the characteristics of magnetic relays is being made by ingenious design. Often the electric supply undertaking will require a specific relay setting on the incoming circuit-breaker, which may result in the time intervals between the operation of different devices in a feeder being very small. Great care is necessary, therefore, in the selection of protective gear to ensure discrimination in operation. When the electrical equipment has been ordered a preliminary protection schedule should be prepared and all relay settings calculated.

Particular attention should be paid to earth fault relays as the fault current level available from the current transformer secondary may be insufficient for operating a trip coil directly. Where a tripping battery, or a voltage transformer supply, is not available, the tripping action can be provided by a manually loaded spring-operated mechanism which is initiated by the current transformer secondary current.

MINING MACHINERY ELECTRICAL EQUIPMENT

The different types of electrical equipment used in opencast mining machinery may be grouped as follows:

(*a*) draglines, shovels, etc.;
(*b*) German brown coal type equipment;
(*c*) conveyor transport;
(*d*) trolley-wire locomotive transport; and
(*e*) miscellaneous pumps, compressors, etc.

The characteristics of the Ward-Leonard system have led to one of the forms of this drive being almost exclusively adapted for the drives in draglines and shovels. It is becoming increasingly difficult to accommodate the physical size of motor generator sets in large machines, and various steps must be taken to reduce shaft lengths, etc. The electrical machinery manufacturer's proposals must be carefully considered to determine the effect on accessibility for maintenance, etc. The method of supporting the bedplates of the motor generator sets should be discussed to ensure the set will remain in alignment if flexing of the excavator frame takes place while operating. Attention must be given to future dismantling of the electrical machinery and facilities for armature, or rotor withdrawal, should be established. To achieve more compact electrical equipment, direct on-line starting of the main motor generator sets may be adopted. Advantage may be taken of the resulting line voltage drop in reducing current peaks, consultation with the electricity supply authorities and the electrical machinery manufacturer being necessary.

If direct on-line starting is not acceptable, 'open transition' auto-transformer starting usually results in simple, less bulky, starting equipment. The present trend is towards the use of static control systems, involving magnetic amplifiers and semi-conducting devices, and this type of equipment is in limited use. As well as there being a reduction in size compared with rotating amplifier equipment, reduced maintenance can be expected.

The electrical equipment of draglines and shovels is adequately described elsewhere.[11,12] As semi-conducting devices of increased current-carrying capacity become available, motor generator sets will be eliminated, with savings in space, cost and maintenance.

The electrical equipment of German brown coal type mining machinery differs from dragline and shovel equipment in that most of the drives are achieved by a.c. induction motors. Space is not so limited, but it is not possible to accommodate all the drives inside the cab as with shovels and draglines. Special care should be taken to ensure that these drive motors are accessible for maintenance. German practice is covered by V.D.E. 0168/1·57* and is described in a number of publications.[13,14,15]

At an early stage after the excavating machinery has been ordered, the grouping of the electrical controls should be discussed with the excavator manufacturer to ensure minimum operator fatigue, attention being given to the physical capabilities of the operating personnel. Ventilation of the electrical equipment should also be considered. Cyclone-type ventilating air filters require little attention, but tend to be bulky. Dry cloth filters give good service if kept clean. A differential-pressure audible alarm should be fitted to indicate choked filters.

Where a conveyor system is planned to ensure that material is not deposited on to a stationary belt, it is essential that the conveyors be started and stopped in sequence; for this purpose speed-sensitive switches, driven by the conveyors and controlling the preceding conveyor, must be installed.

To protect the belt from damage owing to the belt not running centrally, belt-centralizing limit switches must be installed on either side of the belt at any point where rubbing can occur.

To protect the belt from protracted slip, or if the belt breaks, a further speed-sensitive switch should be fitted. The switch is usually set to operate at 90 per cent of full speed. A time-delay relay must be fitted to cover the starting period. Where long modern high-speed conveyors are planned, belt slip must be carefully studied and special integrating devices may have to be adopted.[16] Emergency stopping of the conveyors along their entire length should be arranged. When estimating control cable lengths for stringing along high-speed conveyors, extra must be included for the bights between individual frames.

Where multi-motor driveheads with fluid couplings are used it is essential that each motor circuit be fitted with an ammeter to enable the fluid couplings to be correctly filled for load sharing.

Trolley-wire locomotive units follow conventional lines. So that they can be advanced with the rail track, trolley wires have support standards fixed to the track. Trolley-wire locomotive transport is in extensive use in the German brown coal mines.[17]

LIGHTING

Where operations must be carried out during the hours of darkness good lighting is essential. The problem is not easily solved and each operation must be planned individually. Floodlights mounted on telescopic towers can be advanced without being unstable. Conveyor lighting is simply achieved by fittings mounted on the conveyor structure. For the illumination of working faces lighting fittings can be mounted on the excavating machinery.[18] (The lighting of opencast mines is fully described in another paper.[19])

---

* Specifications for erection and operation of travelling excavators, unloaders, trolleybridges and movable belt conveyors, accessories included, and of mobile traction installations in mines above ground, for open-air working and in similar plants.

STANDBY SUPPLIES

Where essential services such as pumping are required continuously, some form of standby supply must be provided as close as possible to the essential service. It generally impedes mining operations to have standby generators on the floor of the pit and usually a compromise is necessary, the generator being located just outside the excavation.

The high-speed radiator-cooled skid-mounted diesel engine is probably the best choice as a standby prime mover, and can readily be arranged for automatic starting on supply failure.[20,21]

With some mining machines it is possible to 'walk' the machines from the erection yard into the mine by supplying power from a mobile diesel-driven generator with rubber-tyred wheels which is towed by the mining machine. This avoids manhandling trailing cables. In an emergency, such as flooding, a machine can also be 'walked' to safety without the need to provide additional power supply points. This mobile generator can also be used to supply remote sites for drilling, etc.

COMMISSIONING

The commissioning of the electrical equipment must be planned after details of the electrical system are fixed. 'Commissioning schedules' should be prepared, giving detailed step-by-step procedures necessary to ensure that high-voltage equipment may be made live without damaging the equipment or endangering life. The need for commissioning schedules is not always appreciated by engineers not familiar with large high-voltage installations. The importance of systematic checking cannot be overstressed.

The commissioning of equipment, whether it be an individual item or a complete system, is accomplished in four main stages: (*a*) preliminary tests; (*b*) proving tests; (*c*) operational tests (if required); and (*d*) putting into service. The commissioning schedule will be more or less detailed according to the complexity of the equipment being commissioned.

Preliminary tests are those tests necessary during, and on the completion of, construction work to ensure functional soundness before attempting to make the equipment operative. The tests cover the following: (1) tightness of connections and supports; (2) correctness of connections; (3) checking of electrical, mechanical, hydraulic and pneumatic controls; (4) checking of insulation resistances, including insulating oil tests; (5) checking of protective circuit characteristics by secondary injection only; (6) checking of earth resistance; and (7) checking alignment and freedom of bearings, balance, etc.

Proving tests are those necessary to ensure that the whole equipment is electrically sound and in a condition to be safely connected to the power system. The tests cover such items as (1) applied high-voltage tests; (2) electrical checking of phasing; (3) protection-stability and fault-setting tests by primary injection; and (4) performance tests by primary injection (or in the case of a generator, by generation on short circuit or non-commercial load).

Operational tests are those which cannot be made except under system operating conditions and also generally cover any manufacturer's contractual obligations. The tests cover such items as heating runs, efficiency tests and transformer tap-changer operations on load.

Before putting major equipment into service it is usual for the engineer conducting the commissioning to issue a certificate to the operating staff giving clearance for the equipment. After this time, application must be made to the operating staff for permission to work on any item of equipment.

MINING REGULATIONS

The electrical installations in opencast mines are often controlled by two sets of regulations, one being the national electrical regulations and the other that section of the mining regulations concerning electrical equipment. Usually, the national electrical regulations are based on electric utility requirements and the mining regulations on underground practice. During the early planning stages the regulations should be studied carefully and the government departments concerned consulted. Often it is necessary to obtain dispensation from certain regulations and it is essential to ascertain the limitations that may be imposed.

COMMUNICATIONS

The subject of communications is adequately covered elsewhere and only those points affecting electrical planning are considered here.

Modern management requires efficient communication, particularly where mobile equipment has to be controlled. V.H.F. radio-telephone equipment is well suited to this duty in opencast mines, in combination with line telephones for fixed equipment. A central transmitter-receiver station must be located in a position to secure good radio propagation conditions and preferably where it will remain effective throughout the life of the mine.

The provisional allocation of an operating licence should, where applicable, be negotiated with the national authority concerned during the early stages of planning. Some limitations are usually placed on transmitter output and it is necessary to consult the V.H.F. equipment manufacturers to obtain the best arrangements.

The transmission of data, i.e. water-levels, mine output, etc., is of vital importance in a large mine and the remote control of emergency equipment is often required. The three main methods are (*a*) direct wire transmission; (*b*) audio-frequency transmission; and (*c*) ultrasonic frequency transmission.

For short distances the direct wire system is usually the most economical. Miniature components and multi-core telephone-type cables are used, usually with a 50-V d.c. supply.

The audio-frequency system uses the mine telephone system, one pair of wires catering for about twelve channels. A steady signal from an audio-frequency transmitter is received by a band pass filter over the telephone pair. Indication is achieved by interrupting the signal so that 'fail safe' conditions are obtained. For transmission of a variable, a transducer is used to bias the output of a saw-tooth generator. A section of the saw-tooth waveform above the bias potential is amplified and used to switch the transmitter. At the receiver an integrating circuit displays a reading on a meter proportional to the original signal from the transducer. The advantages of using the telephone system cables can be readily seen, particularly where long distances are involved. The cost of terminal equipment for this system is about £60 per channel.

The ultrasonic frequency transmission system is also suitable for long distances. A single-core co-axial cable is used, transmitters and receivers being connected to it in the same way as in the audio-frequency system. Up to 300 channels can be accommodated on one cable, the cost per channel, including cables, being approximately £75.

PERSONNEL

The specialized nature of the electrical equipment used in large opencast mines calls for the employment of electrical engineers of professional standard. The electrical and mechanical departments in most mines come under the control of a chief engineer. A senior electrical engineer should report directly to the chief engineer and be responsible for the electrical engineering aspects of the mine. The functions of the electrical department may be divided as indicated in Fig. 12.

The technical services section covers commissioning, testing, protection, system planning, etc. In carrying out commissioning tests the technical services section must co-operate with the electrical plant section to meet scheduled completion dates, and so on. The major tasks of the technical services section are the co-ordination of protective devices throughout the electrical system, their testing and the investigation of mal-operations, insulation co-ordination, all major testing, system design and planning. The section engineer must be a professional engineer of academic bent, capable of working closely with his colleagues and having full confidence in the long-term advantages of employing his section.

The electrical plant section covers construction, operation and maintenance, and the section engineer should be a professional engineer of wide practical experience, with good organizing ability.

The communications section covers all aspects of communications, including line and radio communications, light current signalling and controls. The extent of the communications system usually determines

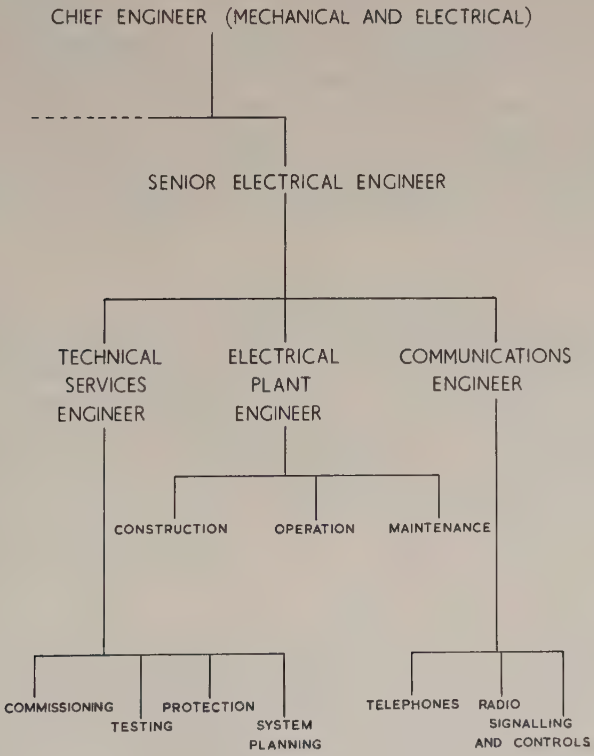


Fig. 12.—Organization of electrical department.

the level at which the communications section engineer is employed, but in most large mines, particularly where assistance from manufacturers is not available, a professional engineer is necessary, while where the communications system is relatively small and assistance is readily available a senior technician may control this section.

For construction work, movement of plant, etc., a team is required. Much of the maintenance and testing work, however, must be carried out by the individual rather than a group of tradesmen under the control of a supervisor. There is, therefore, the need to employ a higher technician–tradesman ratio than in most mining operations. Because of the difficulty often experienced in starting up the electrical engineering division of a new mine owing to shortage of adequately trained engineers, technicians and tradesmen, it is usually necessary to train all grades of staff in their duties.

The engineers, particularly the senior electrical engineer and technical services section engineer, should where possible be engaged at the planning stage. During the development stage, the electrical plant engineer should be employed on the supervision of erection of plant and equipment by contractors and direct labour. Tradesmen and technicians should preferably be engaged during the development stage and be employed upon erection work. Introductory courses should be arranged to familiarize them with equipment and techniques outside their previous experience. By the commencement of the development stage a nucleus of trained staff should be available to take over the electrical side of the mining operation.

CONCLUSION

The electrical system is a vital part of a large opencast mine. To achieve maximum efficiency and economy, careful and detailed advance planning is essential. The size and complexity of the equipment involved is such that there must be complete collaboration between the mining engineer and the electrical engineer throughout the planning and commissioning stages.

Acknowledgements.—The author thanks Powell Duffryn Technical Services, Ltd., for permission to publish this paper and acknowledges the considerable help given by colleagues in preparing the paper. Acknowledgement is also due to those manufacturers who have generously supplied information and illustrations.

REFERENCES

1. Stewart, E. D. J. An introduction to planning of development of brown coal open cuts of the State Electricity Commission of Victoria. *Proc. Aust. Inst. Min. Metall.* no. 194, June 1960, 17–56.

2. Gaffert, G. A. *Steam power stations.* (New York: McGraw-Hill, 1952), 630 p.

3. Carr, T. H. *Electric power stations.* (London: Chapman and Hall, Vol. I, 1954: Vol. II, 1955), 605 p. and 832 p.

4. Westinghouse Electric Corporation. *Electric transmission and distribution reference book.* (East Pittsburgh: Westinghouse Electric Corporation, 1950), chapter 8, 233–64.

5. Westinghouse Electric Corporation. *Electric transmission and distribution reference book.* (East Pittsburgh: Westinghouse Electric Corporation, 1950), chapter 18, 610–42.

6. Hamilton, D. E. Mine power systems. *Coal Age*, **66,** Feb. 1961, 110–14.

7. Westinghouse Electric Corporation. *Electrical power distribution and protection for open pit mines and quarries* (pamphlet).

8. General Electric (U.S.A.). *Power distribution for open pit mines*, by D. Stoetzel, W. D. Heinz, R. B. Bennett, S. C. Cooke (pamphlet).

9. Thielen, H. Die Stromversorgung von Baggerstrossen. *Braunkohle Wärme Energie*, **7,** March 1955, 101–4.

10. Cotton, H., ed. *Electrical equipment in mines.* (London: Newnes, 1955), chapter 2, 24–79.

11. Roberts, E. A. Electrical drives for shovel and dragline excavators. *A.E.I. Engng*, **33,** May-June 1963, 148–54.

12. Gatliff, P. W. R. Electrical drives for excavators. *Engl. elect. J.*, **16,** Sept. 1959, 23–31.

13. Siemens-Schuckertwerke, AG. *Electrical engineering aspects of open cast mining machinery in the lignite industry*, by H. Schulte (pamphlet).

14. Schulte, H. Die Anwendung neuzeitlicher Regelungstechnik bei elektrischen Antrieben von Tagebaugeräten (Application of modern feedback control techniques to electrical drives in opencast mining equipment). *Siemens-Z*, **32,** Aug. 1958, 603–5.

15. Brown Boveri & Cie AG. (Mannheim). *Installation von Fordergeraten im Braunkohle-Tagebau*, by O. Guthmann (pamphlet).

16. Ratnam, G. Electrical aspects of opencast lignite mining. *Proceedings of Symposium on Mechanisation of Mines in India* (Calcutta: Books and Journals Private Ltd., 1962), 30–7; 77. (Special issue of *Journal of Mines, Metals and Fuels.*)

17. Siemens-Schuckertwerke AG. *Elektrotechnik im Braunkohlen-Tagebau*, by H. Pistor and W. Schadow (pamphlet).

18. Thielen, H. Beleuchtungsfragen bei Tagebaugeraten. *Braunkohle Wärme Energie*, **7,** April 1955, 125–34.

19. Bell, W. B., and O'Sullivan, P. E. Lighting of opencast coal mines, with an appendix on the lighting of railway sidings. (Paper 30, this Symposium.)

20. Brealey, S. C. The Neyveli lignite project in India. *Min. elect. Engr*, **44,** Sept. 1963, 71–82.

21. Berman, L., and Woodhead, E. H. Automatic starting and control of diesel engines. *Engl. elect. J.*, **14,** Dec. 1956, 32–8.

Discussion

(Papers 20–23)

Mr. R. N. Pryor,* introducing Papers 20 to 23, said that as the basis for discussion they had four excellent papers.

Paper 20, 'Development of the Kedia d'Idjil orebodies (S.A. des Mines de Fer de Mauritanie) MIFERMA Islamic Republic of Mauritania', had come from France and they were very fortunate to have such a masterly paper on such a huge project by such eminent engineers.

Papers 21 and 22, 'Determination of volumes in opencast mining by aerial surveys', and 'Volume assessment by photogrammetric methods', concerned one aspect of the control of opencast operations.

Paper 23, 'Electrical planning for large opencast mines', covered most authoritatively the one aspect of planning which many mining engineers tended to leave to the specialist; and for that reason it was doubly welcome at the present symposium.

Miferma started in 1952 as a prospecting company and the first ore was shipped in June, 1963. The capacity of the whole installation was for some 6 000 000 t of ore per year; 186 000 000 t of ore had been indicated by drilling, with another 105 000 000 t possible. A railway 650 km in length had been built through very formidable country, involving the crossing of a large area of 'live' sand-dunes as well as a major tunnel, and the port would, in the future, take vessels of up to 100 000 tons. All that had been carried out in an area which initially had practically no water and a summer temperature exceeding 50°C. Wind and dust were ever-present and since 1953 no appreciable rain had fallen. The whole project involved an investment of some £60 000 000. The authors explained how the fantastic operation was planned, how long it took, how many men were needed and what tools they decided to use to carry out the task. Eleven years elapsed between the formation of the Company and the first shipment of ore.

The first five years were spent in extremely arduous conditions on the initial drilling. By that time a report was issued which was favourably received and further finance was forthcoming. The next three years were spent in further drilling and intensive preparation of specifications, designs, programmes, estimates and contractor's terms. Finally, in

---

*Rio Tinto-Zinc Services, Ltd., London.

April 1960, they were ready, the finance was approved and construction started.

Every paragraph of the paper was packed with interesting facts; nevertheless he thought the most fascinating subject dealt with was the organization of planning. The paper told how the planning was carried out in those decisive years 1957–1959. Figs. 10 and 11 (pp. 472 and 473) showed the organization set-up and how it changed as construction proceeded. Figs. 12 and 13 (pp. 474 and 476) showed something of the charts prepared to guide and programme the project. It was interesting to note that it was evidently the construction of the railway which took the major effort and time.

He would assure the authors that everyone was glad to have a record, crammed with technical information, of their magnificent achievement.

The authors of Papers 21 and 22 had described in simple language what was, in fact, a complex precision tool—the first-order plotter from a stereoscopic photo model. That tool, which had been much developed in recent years, was fully at the disposal of opencast engineers for 'cubing' the holes that they were making, as well as the dumps where they were putting the spoil. The authors gave a very clear description of how the tool could be used, together with its limits of accuracy.

He had been fortunate in seeing those instruments being used in the United Kingdom and elsewhere and the number of installations was growing all over the world. Fig. 1 (p. 495) of Mr. Dawe's paper illustrated the instrument.

He would like to ask the authors at what stage it became economic to employ photographic methods. Clearly, if the field measurements and subsequent office work were going to take a ground surveyor four or five months to carry out there was a case, in time alone, for using aerial methods. Equally clearly, if the work would only take the ground surveyor a week there could be but small grounds for a plane flight, photography and the subsequent operations. Somewhere between the two lay the break-even point and he would be very interested in what the authors thought.

Time and cost were, of course, likely to be the prime considerations, but the authors made very clear certain other advantages. (i) All the ground was measured, not just spot positions with interpolation as in ground mapping; (ii) the possibility of transcription errors might be reduced, particularly if the *XYZ* co-ordinates were fed straight from the plotter on to punched-tape or cards; (iii) accuracy might be just as good or even slightly better than conventional methods of cubing.

The principal disadvantages, as given by the authors, were that it was uneconomical to fly small areas, and that weather conditions might delay the work, particularly in cloudy places such as Great Britain.

It was large civil engineering projects which had pioneered the use of aerial photography for cubing, but he was sure that opencast mining would not be slow to make use of the new tool wherever and whenever it was economically justifiable. The two very lucid papers would be of great help to mining engineers in explaining just what assistance was now available.

Paper 23, 'Electrical planning for large opencast mines', provided a guide for mining engineers to the problems associated with electrical planning. Apart from the carefully prepared and detailed information in the text, there was also a very comprehensive bibliography which drew attention to the sources of further information.

It had surprised him to learn that pit distribution voltage might now be as high as 35 000 V, and he would like to ask Mr. Atkinson how large a power requirement a machine must have to make the use of such a high voltage of marked economic advantage.

Fig. 3 (p. 508) gave very interesting graphs showing the power requirements of different capacities of excavator, and reminded them that machines with large motor generator sets usually had synchronous motors which assisted the power factor, while the smaller a.c. induction motors on conveyor belts, etc., adversely affected the power factor unless suitably compensated.

Tables I and II (pp. 510 and 523) gave useful guides for anyone engaged in drawing up estimates of electrical plant.

Mr. J. Chiswell* welcomed the opportunity to direct various comments and questions to Mr. Rawiel and Mr. Dawe.

When considering the problem of volume assessments in the quarrying industry it was always worth while to consider having a topographical plan prepared of the area, as opposed to having one specific question answered. The major cost in air survey was positioning the aircraft, finding suitable weather for the photography and setting up photogrammetric machines. Once those costs had been accepted one must make the best use of the service.

Volumes should be determined by taking vertical sections through the area, as opposed to measuring the contour areas—that system provided a record of the ground sections, which were much more useful for many considerations of planning quarry developments.

If a volume assessment were made from a 1:500 scale plan with a contour interval to suit the topography, he wondered if it were possible to derive any assessment of the accuracy for varying volumes, i.e. percentage accuracy, based on the assumption that the calculations would be carried out by Simpson's Rule.

---

*English Clays Lovering, Pochin and Co., Ltd., Cornwall.

He did not foresee the wholesale introduction of photogrammetry into quarrying practice where there was probably the largest number of individual volumes to be measured—in the field of overburden removal. Any large company engaged on a regular programme of overburden would require a regular monthly survey for purposes of yardage control and payment of the contractors; it would not be economic for an air-survey firm to position an aircraft on such a regular basis and, in any case, the surveyor (or should he say the ground surveyor?) was often called upon to differentiate between the varying types of overburden such as soil, subsoil, normal overburden or rock. The surveyor also assisted in solving any drainage problems associated with bulk earth works; that would not be feasible by an air-survey method.

For all photogrammetric work which was being carried out for a company having a survey department, provided that department had some experience of air photographs, considerable economies could be shown by providing the necessary ground control survey from within that department.

Mr. D. L. Pentz* said he would like to ask the authors of Paper 20 what sort of rock mechanics programme they envisaged and, in that connection, what factors they had considered in relation to the comparison with other mines. There was considerable danger in that method since the geology, tectonic history and thus the pattern of discontinuities were very rarely the same. Also the effect of earthquake or seismic shocks on the stability of a slope and, if the area were subjected to that effect, an allowance for a percentage of 'g' in the analysis should not be ignored. Finally, even though the area was subjected to minimal rainfall, that did not preclude the possibility of water problems in slope stability. He noted that water had been found 50 m below the collars. Thus at the 400-m level in the mine they would very likely have fissure water pressure and a consequent reduction in the shear strength of the rock face.

Mr. A. Shadmon,† referring to Paper 20, asked for additional information about the construction of the port. Further details of the materials used, the quantities involved and major difficulties encountered would be of great interest. Were any special blasting techniques required to obtain the rock material for constructing the quay and what were the sizes required?

The Baie du Lévrier appeared from Fig. 2 (p. 451) to be fairly landlocked and he wondered if any silting problems were envisaged.

---

*Royal School of Mines, London.
†Ministry of Development, Jerusalem.

Mr. B. G. Fish,* referring to Paper 23, said that one particular problem often arose, certainly in the United Kingdom and possibly elsewhere, when items such as junction boxes and isolators were being specified for rough outdoor use in quarries; the difficulty lay in the fact that manufacturers did not design for that specific service, the choice being between light, indoor industrial equipment, designed for factory use, and very heavy, complicated flameproof units for underground mining application. Of those two the former was not nearly robust enough for outdoor service, while the latter, although admirably robust, was of unnecessarily complicated and expensive design for quarry application.

There was a distinct need for a range of equipment which was strong and well protected for the service, but which incorporated a simple and economic design. Could the author comment and perhaps indicate what prospects there were for that requirement being met?

Mr. J. W. White,† referring to Paper 20, said that by any standards the concept was extremely courageous and it was very evident that decisions to proceed had had to be made when the information available was not always complete. There were one or two questions he would like to ask. The authors said that to compete with mines currently being developed, the ore would have to be up-graded from around 58 per cent Fe to about 64 per cent and he wondered if, with recent tendencies, that might still not be high enough. It appeared that the Mauritania ore at present being shipped was run-of-mine ore presumably at about 58 per cent Fe, without any beneficiation. Perhaps the authors would elaborate on their proposed methods of beneficiation to bring the ore to the required competitive standard.

The terminal bins at the railway-loading point were of 800-ton capacity. Was the wheel reclaimer duplicated to assume standby capacity in case of breakdowns?

Had the incidence of Harmattan duststorms caused any lengthy delays to the railway operation by poor visibility or physical covering of the task with drift sand. He noted that provision had been made at the mine for the use of a skipway. Information regarding the size of the skip envisaged and the track gauge of the skipway would be welcomed.

Regarding labour, was it intended to use largely local indigenous labour or to import from Senegal? The ratio of foremen to labour at the mine was as high as approximately one to two; perhaps the authors could explain that high proportion.

---

*Institute of Quarrying, London.

†Sierra Leone Development Co., Ltd., London.

Mr. C. B. Parker*[*] asked if the authors of Paper 20 could give a percentage breakdown of exports to the principal purchasing countries.

Turning to Paper 23, he said he would be interested in the author's comments on the advantages or necessity of using a four-core cable in a ring main. Older mines were fitted with three-core cable ring mains and he knew of an occurrence which could have led to a fatal accident, part of a substation being burnt out owing to the fact that no earth leakage had been fitted. It was suspected that a closed circuit was made through a watercourse to loco rail. E.L.P. was being fitted to cover area groups of machines.

Mr. R. F. Lethbridge[†] said he had had the great pleasure to have been associated with the very beginnings of Miferma. Mr. Choubersky and he had formed part of the original mission to Mauritania in 1951 and he had vivid recollections of the conditions under which early teams had had to work. The present air-conditioning, even of excavator cabs, was in great contrast to the heat in the vehicles used at that time. He was interested to note that their early target of 100 000 000 tons had been largely exceeded and that nearly three times that figure was the present potential.

Although the authors had given the bare figures, they had not brought out the great difficulty that the water supply gave in the early days. The conflicting demands of the nomadic herdsman and the diamond drillers for the small amounts of water made the development stage very difficult.

It was twelve long years from the first mission in 1951 to production in 1963 and he thought the success of the whole operation was worthy of great praise, as was the valuable paper so well describing it.

Mr. D. L. Ward,[‡] referring to Paper 20, asked if the authors could give details of the slurry-blasting techniques and also of the results obtained with, say, AN–FO techniques. If angle drilling had been adopted in the drilling of the benches he would welcome comments on that subject as applied to the project under discussion.

Mr. K. A. Fern[§] said that he was very interested in the use of ammonium nitrate in tropical conditions because of the transition temperature of that chemical. He would therefore welcome any information from the authors of the Miferma project on the behaviour of ammonium nitrate, in storage and in use, in the high-temperature conditions described.

---

[*] De Beers Consolidated Mines, Ltd.

[†] Rio Tinto-Zinc Corporation, Ltd., London.

[‡] Roads Reconstruction (Quarries), Ltd.

[§] Cyanamid International, Ltd.

WRITTEN CONTRIBUTIONS

Mr. G. J. Shrimpton:* I shall be grateful if the authors of Paper 20 will give a brief outline of the initial survey for the harbour, camp-sites and the railroad. Were any reliable maps in existence at the start of operations? I presume that for such an extensive operation air-survey techniques played a prominent part; I wonder, therefore, at what stage it was found necessary to resort to more conventional ground-survey techniques. For example, was it possible to obtain from the air survey the grade, curve radii and computation of excavation quantities? Finally, what percentage of the total development cost of the project was borne by air-survey?

Mr. J. C. Davey:† The Kedia d'Idjil ore deposits of Mauritania appear to have several geological features similar to the large iron ore deposits of the Imataca region of Venezuela, but without its more favourable natural conditions of terrain, propinquity to shipping point, good labour supply and plentiful water.

Comments by the authors on the following points will be welcomed.

Exploratory drilling: The plans (Figs. 4 and 6, pp. 454 and 455) of the F'Derik and Tazadit orebodies show the location of a number of bore-holes, apparently spaced irregularly. After probably some initial probing, or test bore-holes, did drilling to prove ore reserves follow a systematic grid pattern and, if not, what were the reasons for adhering to random drilling?

The reported total indicated reserves range from 39·83 million tons for F'Derik, 79·5 million for Rouessa and 136·93 million for Tazadit, yet the tons per foot of hole drilled average respectively 2900, 24 200 and 27 800. Is there any reason for this apparent great difference in the density of drill holes?

Equipment: The trucks in use are said to be the largest in standard production (60 tons); are the truck bodies of aluminium? Recent experience in large Canadian open-cut mines has shown that not only is there a big reduction in dead weight compared with the more conventional steel truck, but the increased load capacity results in lower operating costs. Other advantages include a higher strength–weight ratio, a lower capital cost and, where sea air or other corrosive atmosphere is involved, the aluminium body is preferable.

---

*Camborne School of Mines.

†Independent Consultant.

Authors' Replies

(Papers 20–23)

Monsieur J. Audibert, Monsieur P. Caruel and Mr. A. Choubersky: In reply to Mr. J. C. Davey (p. 538) drilling first started on F'Derik. The system was laid out according to geological considerations on sections spaced 100 m along the strike. Holes were angled so as to be as nearly perpendicular as possible to the dip and strike. Most sections carried several holes.

The difficulties of the orebody had initially been underestimated and, once the sections had been completed, it was found difficult to correlate them; some cross holes and intermediate holes were therefore put down.

Meanwhile, the iron ore market had changed and it became apparent that the ore would have to be mined at a higher grade than was originally intended and that this would require still more detailed information on the orebody.

A longitudinal tunnel was therefore driven through the orebody about 100 m below mean outcrop level, and the orebody was sampled by a series of diamond drill holes and long drifter holes set to a systematic pattern.

The grouping of holes is more apparent than real, though some holes had to be offset because of difficulties of terrain. This apparent grouping comes from the variations in the directions of dip and strike, particularly at F'Derik.

The cross-section (p. 454) is not typical of the drilling. The holes were fanned out from one emplacement because this was the first section to be drilled and at that time it was very difficult to prepare drill emplacements and approaches with the equipment at our disposal.

The figures on page 459 show that by 1957 the ratio of reserves to metres of development was approximately:

9000 t/m at F'Derik
55 500 t/m at Tazadit } proved and probable
29 500 t/m at Rouessa

The difference between F'Derik and Tazadit is partly explained by the fact that F'Derik was undoubtedly 'overdrilled', while at Tazadit not only was the orebody far simpler and much thicker, but also a lot of experience had been gained. The principal reason for the difference comes from the fact that Tazadit outcrops along a dip slope over a height of 130 m, while at F'Derik the outcrops run along the narrow crest of the hill and the orebody cannot be reached without cutting through a considerable thickness of wall rock.

As regards Rouessa, drilling was suspended in 1955 and only resumed in 1964. Meanwhile, a geological survey, carried out in 1957–58, showed that the outcrops on which the drilling had been based had been very considerably underestimated. The figures for proved and probable reserves are based as much on hillside exposures as on the drilling. They certainly represent a minimum and the initial results of the campaign at present under way give us reason to think that they will be very substantially increased.

To start the operation the company adopted the principle of taking only such equipment as had already afforded ample proof of its reliability by extensive service elsewhere, both as regards operating and ease of maintenance. When the first orders were placed it was considered that that was not yet the case for aluminium in transport and handling equipment.

In reply to Mr. G. J. Shrimpton (p. 538), for many years the Baie du Lévrier has been one of the principal fishing grounds on the west coast of Africa and an anchorage for the French fleet. As such, it was covered by excellent admiralty charts.

Furthermore, in 1953 the French Government carried out a detailed survey of the bay to see whether it would be suitable for a deep-water harbour. These documents were made available to the company.

In 1959 a short hydrographic survey was carried out by the Public Works Department of French West Africa to make sure that the land and sea surveys were properly connected. The land survey was at the end of a long Saharan triangulation carried out independently of the sea surveys and it was felt advisable to make sure that no possible errors had crept in. Simultaneously, a study of tides and swells was begun and continued throughout the period of construction.

Before construction was finally sited, detailed soundings of the area were made and the sea bed was tested by borings.

The only maps available along the railway route were the small-scale Saharan series, quite inadequate for engineering purposes.

In some of the flat areas of the Sahara conditions are such that detailed aerial surveys to a large scale are difficult to carry out and can be misleading. They were found inadequate for construction purposes.

In 1956–57 three engineering companies were commissioned to survey the strip of ground through which the railway would have to pass. Their work was co-ordinated by the Public Works Department of French West Africa.

This general survey was used in 1958 to establish the longitudinal profile of the line and to determine the volume of earthmoving necessary to prepare the construction estimates.

Once the decision to go ahead had been taken, alternative routes through the surveyed strip were determined by the surveyors and the centre line finally chosen was pegged out.

At the mine the French Government, which in 1952 was carrying out a general aerial survey of French West Africa, agreed to survey the Kedia d'Idjil area 'out of turn'. A set of 6 special sheets was prepared to a scale of 1:20 000 and these sheets were used for the first three years of exploration, supplemented, of course, by ground surveys.

The company then had an aerial survey made of all the northern and eastern part of the range which contained the outcrops. This survey also covered a strip some 2 km wide of the plain adjoining the range where the railway terminals, town site, water catchment areas, etc., could possibly be situated.

This survey was made to a scale of 1:5000, except around the major outcrops where the future pit areas would be situated; these were surveyed to a scale of 1:2000.

The surveys were used as a basis for all layout planning and for most of the engineering. Ground surveys were only used where a precision greater than 1:1000 was required. The aerial surveys constituted a very low proportion of the total expenditure.

In reply to Mr. D. L. Ward and Mr. K. A. Fern (p. 537), all blasting is done by vertical holes. Angle holes have not yet been tried.

The holes, which are 12 m deep, were originally loaded with classical explosives (SOFRATEX). These were then limited to the bottom two metres and 6 m of AN–FO were added. The hole was stemmed. The AN was supplied by the 'Office National Industriel de l'Azote'. Some holes were tried out entirely on AN–FO.

In order to improve fragmentation and to increase spacing, TNT and NG slurries were then tried out. Results from the two slurries were comparable. Spacing was increased by 20 per cent and secondary blasting practically eliminated.

Where the AN or the slurries are kept under a well ventilated shelter, no problems of conservation have so far arisen.

Mr. C. B. Parker asked for export figures (p. 537); in 1964 Miferma exported 5 000 000 t of ore, distributed as follows:

	%
United Kingdom	28
Fed. Rep. of Germany	25·5
France	19·5
Italy	15·5
U.S.A.	3·5
Netherlands	3·5
Belgium	3·5
Others	1

Replying to Mr. J. W. White (p. 536), the authors regret that the description of ore grading was not as clear as it could have been. Fifty-eight per cent Fe is an 'assay limit'. Material situated in areas with a grade below this limit, as outlined by the preliminary drilling and sampling, is taken to a dump and set aside for future beneficiation. Only material assaying over 58 per cent Fe is shipped as ore. In 1964 this material averaged 64·7 per cent Fe, the grade being kept close to this average by blending on the stockpiles. This grade was higher than had been expected because, in estimating, a large margin had been allowed for possible pollution on mining.

Reserves of high-grade ore are still very large and the grade could yet be improved by putting the 'assay limit' higher. This would mean that a greater proportion of material would have to be put aside for future beneficiation, which would not constitute a very serious handicap. There exist very large quantities of material suitable for beneficiation which have not been included in the figures of declared reserves and the problem is being studied. The matter is principally one of economics.

Tests carried out to date show that this off-grade material is amenable to beneficiation by several methods and has the advantage of requiring very little crushing. The major problem to be overcome is the shortage of water but, as a counterpart, the run-of-mine ore contains less than 1 per cent moisture and thus lends itself particularly well to dry methods. This is the line that present investigation is taking.

A second wheel reclaimer was not considered necessary on the mine stockpile because trains can also be loaded directly from the main conveyor belt, by-passing the stock. This is a procedure used only in emergency.

During the two years of operation traffic has never been held up on the railway by dust- or sandstorms. The track is regularly inspected for possible build-up of sand, abnormal wear of rails, etc.

No definite skip design has yet been adopted and various alternative solutions are now being studied in order to improve on the original plan.

Hardly any Senegalese labour is now used. Local labour is now put through the company's training centres.

The present high ratio of 'foremen' (or rather 'craftsmen') is due to the very low level of industrial proficiency in the country. The ratio is slowly improving.

Replying to Mr. A. Shadmon (p. 535), a detailed description* of

---

*COURSIN, A. Les installations portuaires de la S. A. des Mines de Fer de Mauritanie (MIFERMA) à Port Etienne. *Construction*, *Paris*, **18**, 1963; no. 6, 241–6; no. 7, 278–85.

the port construction has appeared. Most of the concrete was made with screened limestone coming from the ballast quarry opened up some 60 km from Port Etienne along the railway line. The only special blasting was that used to obtain hard basalt for the prestressed concrete beams.

The Baie du Lévrier is fully protected from the sea but is immune from silting because no river flows into it or has flowed into it. It appears to have been formed at a relatively recent date by the drowning of a topographic depression.

The deep channels seem to correspond to topographic features which reproduce those occurring on land. The only currents are tidal, longitudinal to the bay and parallel to the channels. They have a scouring action and care was taken during construction not to impede them. Hydrographic surveys carried out over a number of years by the French navy show no changes of the sea bed.

In reply to Mr. D. L. Pentz (p. 535), the pits are situated entirely within very hard, compact, siliceous, banded hematite quartzites whose beds dip at an average of 60°.

The shale horizons within the range are all situated well away from the pit area, except for one very inconsistent bed of what is supposed to be a highly altered Basement shale which lies parallel to the footwall of Tazadit some 300 m away. Between this bed and the footwall lie hard, banded hematite quartzites which were explored by the pit and tunnel dug to house the crusher and the head of the conveyor system.

Except for the pockets of 'soft ore', which are sandy in texture, the ore is as compact as the wall rocks.

The only possibility of slippage in the pit might occur if two or more gouge filled fissures were to intersect behind a face and swelled after rain of sufficient intensity to penetrate them.

Such fissures are practically impossible to observe from the surface, which is mostly covered by broken slabs of quartzite embedded in windblown sand. In drill holes they pass unnoticed as the clay is washed out, leaving a residue of fragments of ore or rock.

When planning the pit, therefore, a safety bench 10 m wide was left every 20 m on the hanging-wall, where the beds dip outwards, and every 10 m on the footwall, where the beds dip inwards. This conforms with usual practice in mines similar to Tazadit.

Now that faces have been opened up they are surveyed in detail by the geological department after every blast and plans and sections to a scale of 1:500 are kept level by level. Among other things, all faults, fissures, passages of clay, etc., are noted. They are not very frequent.

Pit faces, benches, etc., are also regularly inspected. If dangerous jointing or a tendency to slip were noticed, the area in question would be consolidated.

The matter is being carefully noted but there still remain a couple of years before the pit begins going down. For the time being it is still slicing off the top of the hill on which the outcrops occur, and wall overhangs are localized and low.

The Precambrian Shield within which the Kedia d'Idjil is situated is well outside any earthquake zone. The nearest is that of the Moroccan Atlas, 1000 m away as the crow flies.

The Kedia d'Idjil constitutes a sedimentary inlier whose roots are pinched within the gneisses of the Basement complex.

The water it contains is entirely localized within well defined areas of high fissuring which follow the lines marked 'faults' on the plan (p. 453). (An error was overlooked on this plan and the easternmost fault, situated immediately below the title 'Pit Offices', should be pivoted 90° around its point of intersection with the road.)

These fissure zones give rise to spectacular cañons, generally with vertical walls, sometimes over 300 m high. They were cut before the climatic change which occurred in the area.

All these zones were carefully explored by hydrogeologists, and in most cases by drilling, during the search for water. They do not appear to communicate except where two such zones intersect. They are separated by areas of compact, impervious material, folded but little fissured.

The total reserves of water contained in these pockets are not known. At present a total of 500 m^3/day is pumped from three of these zones, with a very slow drop in level. Attempts to increase pumping resulted in a sharp drop and were discontinued. That is why 500 m^3/day are also brought in by rail from the pumping points near Port Etienne.

So far, even moderate rain has been sufficient to re-establish the levels. This occurs very rapidly and can better be explained by the intense runoff trapped in the alluvials at the bottoms of the cañons and penetrating the underlying bed rock than by lateral feeding through underground fissures.

The company would welcome a season of heavy rainfall before work progresses much further. In 1952–53 there was no opportunity of obtaining more than general impressions of the results of heavy rainfall. These impressions were supplemented by expert advice from specialists with experience of other arid areas, in particular the Sahara and Southern Morocco, where mining has been under way for several decades. All precautions which appeared reasonable were taken, but all would be glad to have them confirmed by practical experience.

The rains, when they do come, fall in brief torrential cloudbursts. During a particularly rainy season these may occur every two to three days. There is no topsoil to hold back the water and runoff and evaporation are intense. An hour after the rains the range, except for a few hollows and water holes, is as dry as before.

The fissure areas described in this reply are all remote from the pits and it appears very improbable that fissure water will be encountered when the pits go down.

Mr. R. F. Rawiel: In reply to Mr. R. N. Pryor's remarks on the economy of aerial surveys (p. 533), when considering this question as applied to volumetric assessments in opencast mining, three main factors have to be taken into account jointly—costs, time and instant information.

The basic costs of aerial surveys are naturally higher than ground survey costs and when comparing the cost figure alone aerial surveys would become economic only where extensive surveys are required. It would be difficult to state exact figures for the comparison between aerial surveys and ground surveys, as the shape of the mining quarry, the distance from the air base, prevailing weather conditions and other factors must be considered.

The saving in direct costs is probably not quite as obvious as the improvement of the information obtained from aerial photographs. Once photographs have been secured the assessment of volumes can be carried out in a comparatively short time, independent of conditions at the mine and, provided weather conditions do not prevent aerial photography, the state of mining operations at any specific instant can be preserved for subsequent analysis.

Mr. T. Atkinson: referring to Mr. R. N. Pryor's question (p. 534) on the use of pit distribution voltages as high as 35 000 V, there is no particular power requirement which is associated with any voltage. The decision to adopt a particular voltage is both an economic and technical one. The factors involved are distance over which the supply has to be transmitted, size of the load, voltage regulation, system design (considering the mine as a whole and not just one individual machine), availability of equipment, physical size of cables, etc. Each alternative must be costed before a decision is reached. If a single load of 10 000 kVA, supplied at 11 000 V, is considered, the overall diameter of cable used would be such as to prevent it from being seriously considered as a trailing cable, and a higher voltage would almost certainly be more economic.

I sympathize with Mr. B. G. Fish in his efforts to obtain electrical equipment suitable for quarries (p. 536). Although there is high voltage equipment suitable for opencast mines and quarries, the demand for quarry equipment is probably too small to give the necessary incentive to manufacturers to develop equipment of functional design. One manufacturer in the U.S.A. is producing packaged units which are suited to American conditions and mining equipment, and a British manufacturer offers a robust industrial unit described as suitable for quarries. The equipment best suited to quarry operations I have seen in the United Kingdom was dockside electrical equipment which the quarry owners had modified to meet their requirements. Undoubtedly, equipment of functional design is the solution and there is need for development of this line.

In answer to Mr. C. B. Parker's query regarding the use of four-core cables (p. 537), there is usually the need to provide some form of earth continuity conductor as well as local earth points. With unarmoured cables a fourth core is essential as the cross-sectional area of the screening wires is much too small to carry fault currents. An often used rule-of-thumb formula is that the earth conductor should be not less than half the cross-section of the power cable. This, however, is only a rough guide and the earthing system design should be given the same degree of importance as the power system. The mode of operation, earth resistivity, contact voltage factors, etc., must be considered. Where the regulations permit there is merit in the use of earth-fault current-limiting devices and the use of sensitive earth-fault protective relays invariably results in safer operation. These relays must, however, be checked regularly by qualified staff since they may be required to operate only once or twice in the life of an installation, and the fact that they may be inoperative would not otherwise be apparent.

Following the formal discussion at the symposium, the use of electric trolley-wire locomotives was raised—particularly concerning the safety aspects—and the following points may help to illustrate their operation.

On the Continent there are many installations where large tonnages are transported long distances by standard-gauge, trolley-wire, locomotive haulage. The loading point must be specifically designed for this operation and the excavator operation co-ordinated with the transport system. Where the trolley-wire must pass under the loading point some form of shielding is necessary to avoid accidental contact with the trolley-wire and damage to it. The trolley-wire is usually diverted to one side of the track at this point.

Roads for tracked and rubber-tyred vehicles should be marked out to prevent accidental contact with the live trolley-wire. Where maintenance work is being carried out near live conductors, these should be isolated and the maintenance workers controlled by an access permit system.

The trolley-wire locomotive is generally considered to be most suitable for large opencast mines. The operations must be carefully planned as there is not the same degree of flexibility as is provided by rubber-tyred vehicles. The trolley-wire locomotive becomes economic where long hauls involving large volumes must be dealt with, such as in overburden transport in German brown coal mines.

GROUP IV—OPENCAST MINING
(bedded deposits and general questions)

Session 7—Stripping and Restoration (Papers 24–27)

Chairman: Mr. G. C. Payne *Rapporteur:* Professor J. H. Taylor

622.271.4
621.879.2:622.271.4

Paper 24

Stripping of Rock Overburden

T. M. DOVER, A.C.S.M., M.I.M.M.
Group Manager, Opencast Mines, Ore Mining Branch of The United Steel Companies, Ltd.

SYNOPSIS

The paper deals with problems which arise when stripping overburden, a large proportion of which is rock so hard that blasting is necessary before it can be excavated. In general, methods suitable for hard rock can deal with softer material, but not vice versa.

Suitable excavating machines for hard rock are discussed. The single-bucket machine is at present the only practical type, and working costs usually favour the face shovel. For applications where output is limited and long reach is required the dragline can operate reasonably efficiently if blasting is well carried out.

Methods of transporting spoil are considered, including rail haulage, belt conveyors and trucks.

The problem of selecting stripping machines for cut-and-fill working is outlined and the importance of correct size to give the lowest operating cost is emphasized. Limited ore reserves or outputs may make the most efficient machines for handling material financially uneconomic.

Recording instruments for excavators and recent developments in their design are mentioned.

Geological formations which produce hard rock overburden can be divided into three categories:

(*a*) the covering of a massive deposit of considerable depth and lateral extent, the working of which will take so long that most of the overburden must be dumped outside the area of the deposit;
(*b*) the hanging- and footwalls of veins and masses which must be stripped to prevent their falling in on the workings as the ore is removed; and
(*c*) bedded deposits of comparatively small depth so that an area of worked-out ground is soon made available and into which overburden can be dumped from adjacent unworked areas.

In the case of (*a*) and (*b*) any rock can form the overburden, while in (*c*) it usually consists either of sediments or lavas.

The problem of finding the most suitable machine for stripping in the case of masses and veins becomes one of finding the most convenient loading machine for vehicles or conveyors, and for hard rock, with which this paper is concerned, this must be a single-bucket excavator or possibly a machine of the gathering arm loader type, although the writer has not heard of this application.

Where outputs of stripping are comparatively small and capital equipment must be limited the tractor shovel on either rubber tyres or tracks can be effective as an alternative to a face shovel (Fig. 1), but if large outputs are required the face shovel cannot yet be bettered for low cost and reliability. The limited dump height of face shovels above the base on which they stand makes it necessary for the haulage vehicles to run on the same level as the shovel or only a few feet above it (Fig. 2). A hard bed of overburden over a soft or disturbed bed, for example, may cause difficulties as it may be impossible to build satisfactory haulage roads on the soft bed.

To avoid this trouble a dragline standing on the top of a bench in the hard overbuden may be a satisfactory solution, the longer digging cycle time than for a shovel being compensated by the dragline's ability to carry a larger bucket. The amount of blasting required with draglines is more critical than with shovels since the lump size must not be so great that the bucket is thrown over on striking a lump, but at the same time overblasting may result in the benching from which the dragline must work becoming unstable.

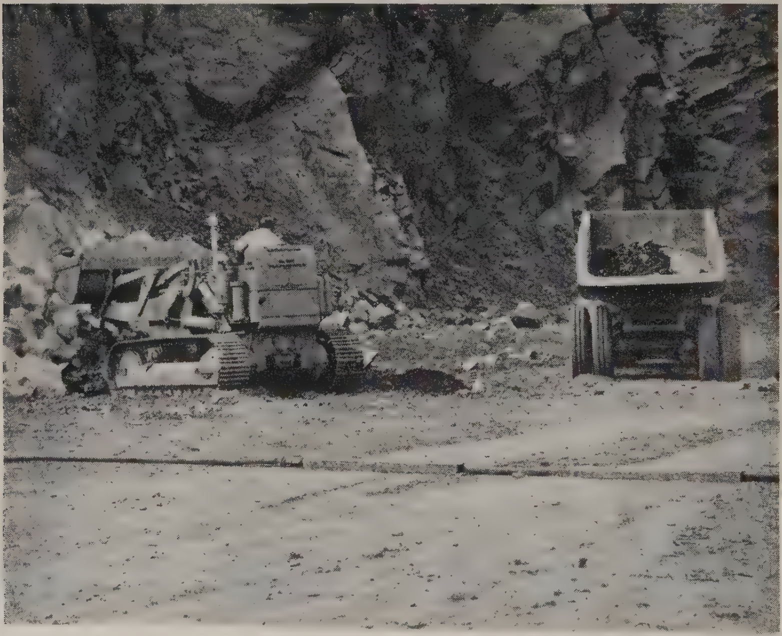


Fig. 1.—Tractor shovel loading rock.

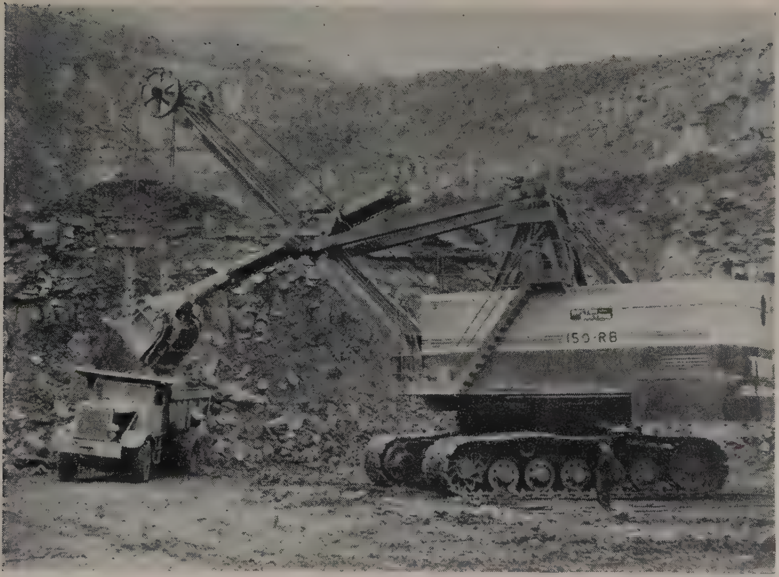


Fig. 2.—Face shovel loading into rubber-tyred trucks.

Draglines can load into vehicles, or hoppers for belt conveyor loading, but the free-swinging bucket and movement of the bucket away from the machine when dumping make ‘spotting’ the bucket over the vehicle or hopper slower and may result in more spillage.

Where it is possible to use it, the close-coupled face shovel is the best machine for loading hard rock overburden.

TRANSPORT OF SPOIL

The transport of rock from the loader to the dump can be by rail, belt conveyor or rubber-tyred trucks.

Rail haulage

Rail haulage, although used extensively in the past, cannot usually compete with other forms of transport for this work and it seems unlikely that any new installations will be made. The high capital cost of track and rolling stock and the high cost of track maintenance when track cannot remain for long in one place are its chief disadvantages.

Belt conveyors

Belt conveyors are probably the cheapest method of transport over short distances, provided the tonnage to be moved is sufficient to defray the heavy capital cost, and the rock is broken small enough not to damage the belts and does not contain material so sticky that it sticks to the belt at the discharge end or at transfer points. In this connection the rate of advance of the face in opencast workings may be critical, as some rock is suitable for conveying when freshly dug but soon weathers to a sticky clay. Seasonal operation to avoid periods of heavy rainfall can also avoid this trouble. A portable grizzly may be necessary at the feed point to prevent oversize rock damaging the belts, and crushers to deal with occasional oversize have been suggested.[1] The impact or hammer-mill type of crusher would seem most suitable because of its moderate weight and its ability to handle a large reduction in a single stage.

Truck haulage

Rubber-tyred trucks appear to suit more varied conditions and are more flexible than any other type of equipment. They have been built in many forms and sizes with many power units. For major schemes the most economic capacity for working with a given loading machine for average haul distance is that which ensures that the trucks are not standing to be filled longer than three or four complete cycles of the loading machine and that part-filled bucket loads are not required; nor should the excavator be waiting for an empty truck. If new equipment is being installed an excavator of the size required to give the output can be chosen and this will indicate the truck size required to carry not more than four times the excavator bucket capacity. The number of trucks is dependent on the round journey time including loading and unloading. With very long hauls cost of haulage may outweigh the cost of loading so that larger vehicles may be economic. Where the discharge point is over the edge of a bank and advances as the bank is built up, the rear-dump type of body is best for rock; side- and bottom-dump types being suitable for fixed-dump stations and small material.

For poor haul roads with soft or slippery surfaces and steep gradients four-wheel drive (4 × 4) or six-wheel (4 × 6) or (6 × 6) may be necessary but, if possible, gradients should be reduced and surfaces improved so that conventional rear-wheel-drive four-wheel vehicles (2 × 4) can be used with a considerable reduction in maintenance, capital and running costs. With good surface and gradient conditions semi-trailers give greatly increased capacity, loads of 100 tons or more being possible. Gradients flatter than 10 per cent can be negotiated under almost any conditions by (2 × 4)-drive trucks.

The introduction of the electric wheel drive, an electric motor and reduction gear built into the centre of a rubber-tyred wheel, is a development which may make very much larger vehicles practical. Power units are generally one or more diesel engines driving through mechanical transmission systems with or without hydraulic torque converters. Possible developments being considered and tried are electric and hydrostatic transmissions and gas-turbine power units. Road speeds up to 35 miles/h are possible, but road construction and surface must be good if such speeds are to be reached.

Cut-and-fill

With bedded deposits there is generally more scope for cut-and-fill working than with masses or veins. Cut-and-fill has the advantage of giving the shortest possible distance for the movement of overburden between cut and spoil bank.

There is at present no practical alternative to single-bucket excavators for hard rock cut-and-fill and thus the face shovel or dragline employed should have sufficient reach to dump the spoil into the worked-out area without intermediate rehandling. In the past transporters have been used to enable a comparatively small excavator to dump far enough away to work deep overburden, but although the system can be made to work satisfactorily, costs are excessively high and output limited compared with the use of a long-reach face shovel or dragline.

EXCAVATING MACHINES

Stripping shovels

The large stripping shovels were developed in the Middle West coalfields of the U.S.A., the largest machine so far built being the Bucyrus Erie 3850B supplied to the Peabody Coal Co. (Fig. 3). This machine has a 115-yd^3 bucket with a maximum dump radius of 218 ft and a working weight of 13 900 000 lb and takes 12 250-hp motor generator sets to supply the power.

The largest stripping face shovel outside the American continent at the time of writing is believed to be the 17-yd^3 140-ft dump radius Marion 5323 machine at the Ore Mining Branch of the United Steel Companies, Limited, opencast mines at Colsterworth in South Lincolnshire, England (Fig. 4). Although the demand for ore has not been sufficient since its installation to allow continuous operation for extended periods, all-in costs of 10d. per cubic yard including depreciation have been reached.

Some particulars of this machine are: cycle time 45–60 s depending on angle of swing; working weight 2 715 000 lb; ballast weight (punchings) 755 000 lb; levelling on four automatically controlled hydraulic jacks; bearing pressure 60 lb/in^2; cutting radius maximum 148 ft 6 in.; dump radius maximum 140 ft; rear end radius 37 ft; bucket bail pull maximum 185 000 lb; motor generator set motor 900 hp supplied at 3300 V.

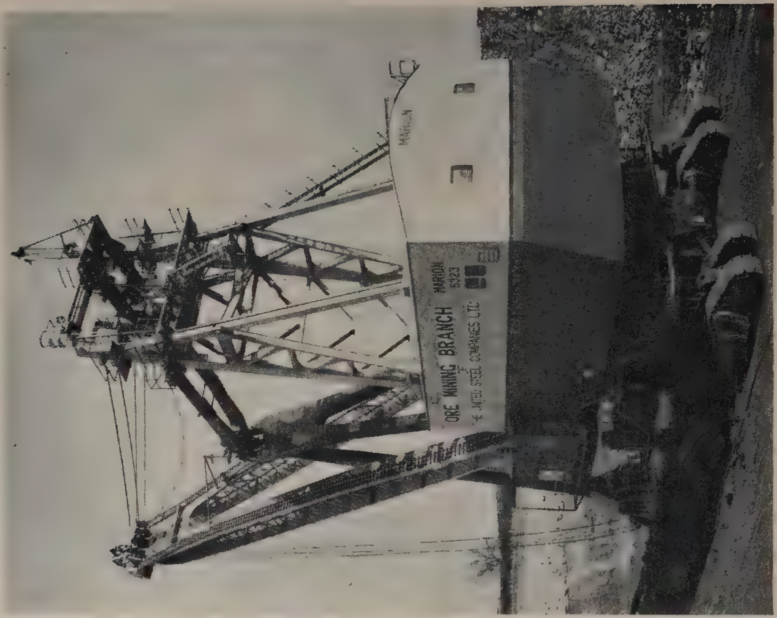


Fig. 4.—Marion 5323 face shovel at United Steel Cos., Ltd., Colsterworth North mine.

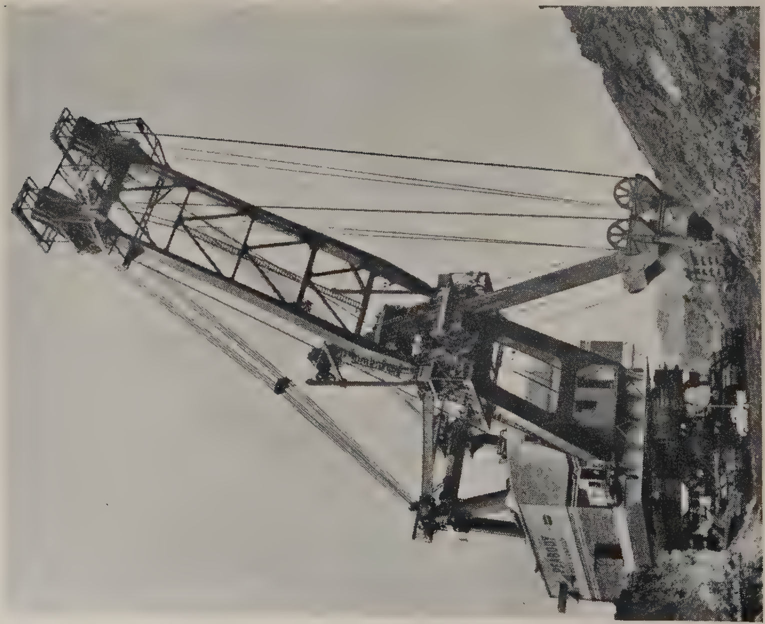


Fig. 3.—Bucyrus Erie 3850B, largest machine so far built.

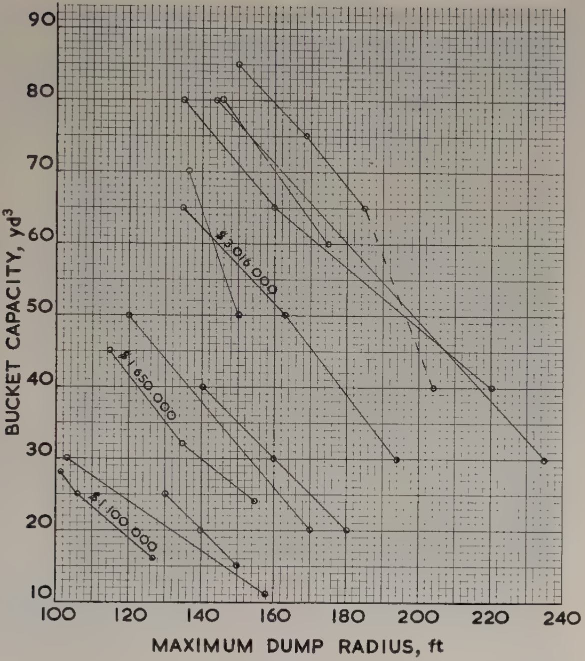


Fig. 5.—Standard stripping shovel duties.

The maximum demand $\frac{1}{2}$-hour rating has been 400 kVA with a power consumption of 320 units per operating hour.

This machine is working in heavy limestone overburden at present 36 ft deep increasing to 55 ft. Its depreciation is calculated over 15 years and it has recorded 82 per cent working time over two shifts. The cost of such a machine is over half a million pounds. A recent check of output confirmed the manufacturer's claim of nearly 600 yd^3 per operating hour in-place measurement. A simple recorder registering the operation of the dipper trip on a time scale is fitted to this machine and has proved most useful in helping to reduce stopped time.

Fig. 5 shows in graphical form the dump radius and bucket capacities of the standard stripping shovels offered by the three known manufacturers of this type of equipment at the time of writing. Prices quoted in 1962 for three sizes of machine in the U.S.A. are shown against the duty line on these machines as a rough guide to capital requirement.

Stripping draglines

When the output required from an ore deposit with medium depth of overburden, i.e. from 40 to 100 ft, is limited, a large walking dragline can be economical in hard rock if reach is important, and its ability to spread the spoil, which makes levelling of the spoil dumps easier, is an advantage. At the Exton Park quarry of the Ore Mining Branch of the United Steel Cos., Ltd., a Ransomes & Rapier type W.1400 has been digging overburden composed of 60 per cent limestone since 1956 (Fig. 6). The main dimensions of this dragline are: cycle time with 90° to 110° swing average 49 s; working weight 3 750 000 lb; no ballast required; dump radius 260 ft; maximum depth of cut 100 ft; bearing pressure on base 14·5 lb/in.^{2}; width overall 73 ft; tail radius 68 ft 6 in.; bucket capacity 20 yd^3; the two motor generator sets are rated at 1500 hp each, supplied at 6600 V.

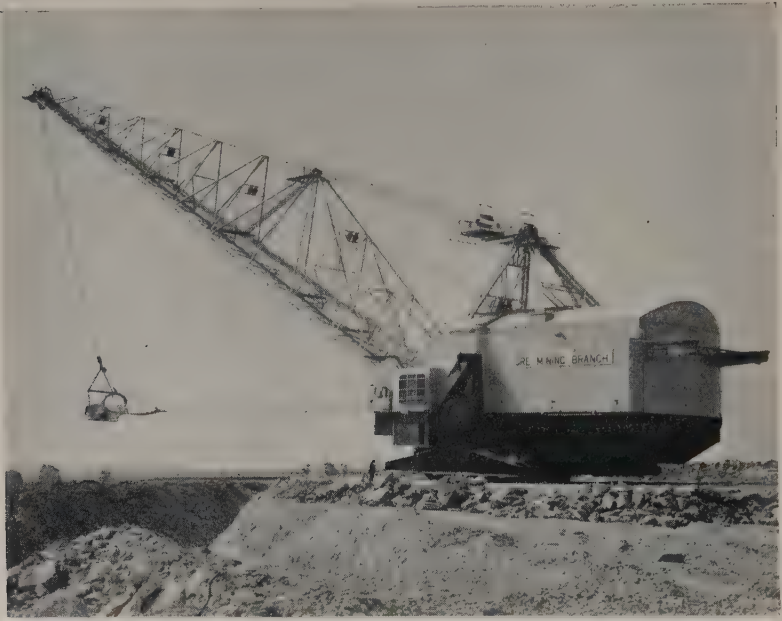


Fig. 6.—Ransomes & Rapier type W.1400 at United Steel Cos., Ltd., Exton Park quarry.

The requirements here which influenced the choice of a dragline rather than a face shovel in hard rock were the need for complete restoration of the surface after working and an output well within the machine's capabilities. Messrs. Stewarts & Lloyds (Minerals), Ltd., have several large draglines ranging from a Bucyrus Erie 1150 B. with 205-ft boom and 19-yd^3 bucket to a Ransomes & Rapier W.1800 with 260-ft boom and 30-yd^3 bucket now under construction. These machines are all digging limestone requiring blasting as part of the overburden they are removing. The world's largest dragline is believed to be the Marion 8800 now under construction for the Peabody Coal Co. This model can carry buckets ranging from 60 to 100 yd^3, with boom lengths from 225 to 300 ft.

Fig. 7 shows graphically the range of duties of each standard model made by the three firms regularly building walking draglines.

To prepare a rock face for dragline working the blasting must be sufficient to ensure that there are few lumps too large to enter the bucket easily, while the benching must be sufficiently stable for a large machine to work and travel on without danger of collapse; this generally means a high burden to retain the charge and placing the charge only in those strata likely to yield large lumps. Horizontal, vertical and inclined blast-holes may be used for this purpose.

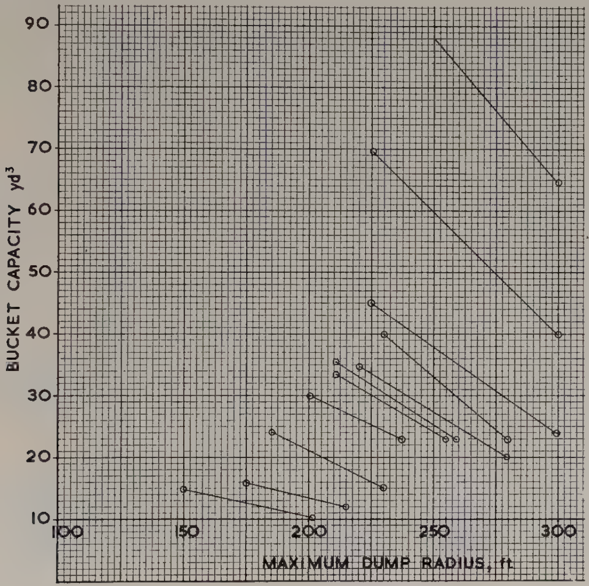


Fig. 7.—Stripping dragline duties.

The plates on the bottom of the tub of a walking dragline, that is the circular base on which the machine rotates when working, are easily distorted and may be punctured by high point loading, and the tub frame may be damaged by uneven loading. For these reasons, when walking draglines work on rock, care must be taken that no large lumps lie near the surface and that the top is as level as possible before the machine travels on to it. Loose small stone can cause trouble by being scraped up by the trailing edge of the tub when walking, forming a ridge on one side and so destroying the level surface prepared to spread the weight of the machine. Wet and frozen clay can also be troublesome through building up a cake on the tub bottom which can make travelling impossible, and a wet clay surface can become so slippery that the machine will slide or turn on its tub. Machines have for this reason fallen over the edge of the benching. The layer of clay or soil used on the surface of hard rock to form a cushion for the tub should be about 12–18 in. Too thick a layer may cause the difficulties mentioned above.

Dragline buckets vary considerably in their digging ability in broken rock, the heavy wide-mouthed bucket generally giving the best results.

SELECTION OF MACHINES FOR CUT-AND-FILL STRIPPING

In selecting machines for opencast stripping it is not always possible to obtain the ideal excavator in type, dump radius or bucket size. Available idle machines or the nearest maker's model often have to be used although not exactly suitable to the application for the work, and conflicting requirements for peak and average output may make it necessary to compromise on the capacity of the machine.

The ore deposit having been prospected and its limits determined, the rate of extraction will be decided, taking into account the market for

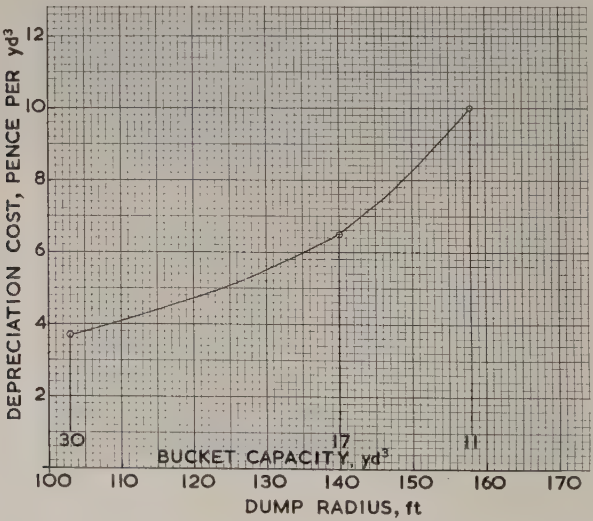


Fig. 8.—Effect of dump radius and bucket capacity of large single-bucket excavator on depreciation cost per yd^3 in place.

Depreciation per hour = 3420 d.; Output per hour with 17-yd^3 bucket = 530 yd^3.

the product over the life of major machines, say 15 or 20 years, or less if the deposit will be worked out sooner. From this rate of extraction, which may vary during the stripping machine life, an expected rate of stripping yardage can be forecast and this must be kept as constant as possible throughout the machine life to obtain lowest cost. Plotting this as a graph of cubic yards per annum may show peaks which can be reduced by bringing in extra machinery in the form of hired plant over the period of the peak or rearranging the method of working to even out the output required. Having determined the output required from the stripping machine during the period of maximum production the appropriate bucket size can be found.

Bucket capacity

The required bucket capacity may be found by the following formula, which should cover the worst expected operating conditions:

$$B = \frac{100.R.O}{FC}$$

where O = output of ore in yd^3 required per hour (if in tons convert to yd^3, for ironstone divide by 1·7)
R = ratio of depth of overburden divided by depth of ore
F = bucket factor, i.e. actual yd^3 moved per hour divided by nominal bucket capacity in yd^3 × cycles per hour (this in broken rock may be as low as 50 per cent)
C = average cycles per hour available for working. Cycle times rarely average better than 60 per hour, and availability higher than 75 per cent of manned hours is difficult to maintain. American operators[2] claim faster cycle times and over 80 per cent availability
B = nominal bucket capacity in yd^3

Dump radius

Dump radius is best determined by drawing pit sections to show the layout of stripping and loading operations. It is generally advisable to adopt the layout which shows the lowest requirements in stripping machine dump radius. As an example of the effect of dump radius on costs, the Marion 5323 with a 17-yd^3 bucket and 140-ft dump radius had a depreciation cost of 6·5d./yd^3 over a given period. This machine is offered with a 30-yd^3 bucket at 103-ft dump radius or 11 yd^3 at 158 ft dump radius. With a 30-yd^3 bucket depreciation cost on the same basis as above would be 3·7d./yd^3 and with an 11-yd^3 bucket the cost would rise to 10d./yd^3 (Fig. 8). Information which will be necessary to draw these sections is the angle of repose or slope to which the spoil will run, 5 horizontal to 4 vertical being a commonly accepted figure for rock, or 39° from the horizontal, while 32° has been measured when fines are present. The swell is shown by the area of the spoil bank section compared with that of the cut sections. If no local information is available 35 per cent can be used for this swell. The width of base of the cut and spoil bank area will be controlled by the ore-loading machine cut width.

Cut faces are generally not as near to the vertical as they appear and allowance for their inclination must be made. Nor can machines travel safely near the edges of open faces and room for cables and for other services may have to be provided.

A decision will have to be made on the maximum depth of overburden the stripping machine is to be required to handle, since this will control both its bucket size and dump radius. It may not be economic to install a machine to handle the maximum depth if this represents only a small proportion of the total yardage to be handled during the machine's life. In this case high and low machine duties for the deposit can be determined and plotted on the graphs of excavator duties (Fig. 5 for face shovel applications; Fig. 7 for a dragline). Where the line joining these high and low required duties cuts the individual excavator model duty line this gives the most suitable bucket size and dump radius for the machine.

The individual excavator model duty lines have been drawn by taking manufacturers' figures for a large bucket at small dump radius and small bucket at large dump radius and joining these points. Improvements in design sometimes enable higher duties to be offered when the minimum figures are known, but this method will enable preliminary investigations to proceed. More detailed investigations of the machines having the required capacities can then be made.

It cannot be too strongly emphasized that although the machine chosen must not be too small either in bucket capacity or dump radius to work through the deepest overburden to be handled without assistance and giving the required output, the installation of a machine too large for the average conditions will lead to much higher costs.

INSTRUMENTATION

It is very helpful to the management responsible for planning new installations of large excavators if factual records of operating times from existing machines in similar conditions are available. Two types of continuous recorders have been used at Colsterworth for this purpose. The first on a walking dragline—and the more elaborate—consists of a continuous strip chart marked longitudinally in angles from 0 to 360° and transversely in hours and minutes. This passes under the recording pen at the appropriate speed. The pen records the angle of swing of the machine in one direction only, a clockwise rotation

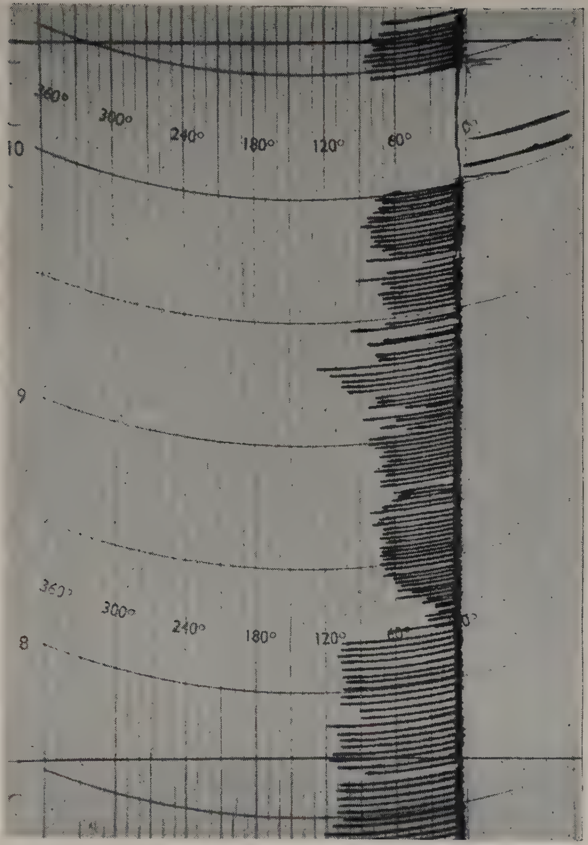


Fig. 9.—Swing recorder chart from W.1400 Ransomes & Rapier dragline.

being recorded and the mechanism returning to zero with any anti-clockwise movement of the machine. The instrument operates through a transmitting 'magslip' driven from the slew gear and a slave 'magslip' in the instrument. (A 'magslip' is an electrical positional indicator operated from an a.c. supply, the shaft of the slave unit taking up the same angular position as the shaft of the driven unit ±1° or less, so that only electrical connections are required between the two units.) A further refinement is a switch on the travel gear which is closed for each revolution of the walk shaft. This causes the pen to move off the slew angle portion of the chart so recording travelling time and distance, each blip representing one step or 6 ft moved. Fig. 9 shows a section of the chart produced.

The simpler recorder is fitted to the stripping face shovels. This has a circular chart making one revolution every eight hours and the trace of the stylus point moves in towards the centre so that three revolutions or 24 hours' working are recorded on each chart. A battery-powered circuit is closed by a micro switch every time the contactor for the dipper trip motor is operated. This moves the stylus giving a blip for each bucket emptied. Although not giving as much information as the swing recorder the simpler version has been found quite satisfactory and invaluable for the basis of work study investigations. The chart speed of recorders should be such that individual cycles can be counted.

DESIGN OF EXCAVATING MACHINES

Although the design of excavating machines is outside the scope of this paper it may be of interest to mention some recent developments.[3]

Air-operated controls lighten the operator's work and reduce the slowing down of working towards the end of the shift through driver fatigue on the smaller machines. The larger machines are generally electric and have very light controls in any case. Ward-Leonard systems of control in excavators have developed with the introduction of rotating control such as the Amplidine and more recently 'static' control using magnetic amplifiers and semi-conductor devices which have no moving parts, giving greater reliability, faster response to the operator's controls, and reducing maintenance.

A walking dragline with automatic swing acceleration and deceleration control is under construction. Although the value of this has yet to be proved, the intention is to reduce the swing time to the minimum.

Maintenance has been reduced by better machining, hardening and finishing techniques and the use of more suitable materials, also with the use of more ball- and roller-bearings and better casings to exclude dirt from moving parts.

Front-end equipment on shovels tends more towards single dipper handles with rope crowd operating from the machinery deck or the Marion system of having the crowd machinery on the 'A' frame. Both have the advantage of reducing the weight on the boom which must be counter-balanced. Where a shipper shaft is used the boom may be braced back to the 'A' frame at this point and the boom beyond jointed to allow for some movement when digging.

REFERENCES

1. Schellhorn, H. W. Crawler-mounted bucket-wheel excavators. *Ore mining and materials handling:* Papers and discussions from the 47th Iron and Steel Engineers Group meeting held in Metz, France, 10–14 June 1963: Special report 82 (London: Iron and Steel Institute, 1963), 10–20.

2. Geissel, R. L., *et al.* Performance of large stripping equipment: a report of AMC Committee on Strip Mining. *Min. Congr. J.*, **47**, March 1961, 77–9.

3. Major-Stevenson, A. Excavator design and practice for pits and quarries. *Quarry Mgrs' J.*, **47**, June 1963, 227–38.

621.879.29:621.879.34:622.271.4 *Paper 25*

Economic Comparison and Detailed Discussion of Dragline versus Shovel for Removal of Overburden

R. W. ZEINDLER, B.Sc.
Export Manager, Marion Power Shovel Company, Marion, Ohio, U.S.A.

SYNOPSIS

In an attempt to clarify the application of shovel and dragline the paper presents both types of operation. Comparison is made with typical sections, production and cost estimates by means of two general classifications of operation and equipment, one being the small producer with relatively small equipment and the other the large producer using large equipment.

The dragline's particular advantages are that longer reach and range are obtained, the unit is more flexible with regard to variations in the configuration and general geology of the deposit, it is more easily manoeuvred than the shovel and for the larger size of equipment considerably lower bearing pressures are possible.

The shovel, on the other hand, has the advantages of greater capacity, positive control of the dipper resulting in better filling of the dipper, particularly in the harder or rockier type of overburden, generally lower cycle time and, as a result of these factors, a lower overall cost per cubic yard of overburden removed.

Present-day mining equipment is extremely varied and is becoming increasingly complex. Much has been written, discussed and argued concerning the most efficient application of two types of opencast mining and stripping equipment—the shovel and the dragline. Little has been resolved.

In any new development or expansion of an existing mining property, economic factors are obviously all-important. On operations requiring the use of large-scale equipment it may well be that capital investment in an individual item of equipment represents one of the largest single investments. A good example of this is the recent occurrence in the stripping industry of the United States of America where draglines and shovels, each costing upwards of $7 000 000, will be employed. Extremely detailed analysis of the overall economics must be made before investing such vast sums of money in a single piece of equipment or individual 'tool'.

Coal, phosphate and iron ore are three types of minerals obtained by opencast mining and, more specifically, by stripping equipment removing the overburden. Any mineral can be mined economically by this method as long as the general geological characteristics are such that an efficient application can be effected. Coal probably represents the one single mineral value that is more widely recovered than any other. For coal, as with other deposits, the total reserves available to a corporation or individual provide the criteria for the size of equipment and resulting capital investment.

With the availability of 200-yd^3 shovels and 100-yd^3 draglines, and the general rule of thumb that the larger the unit the greater the saving, many corporations may feel that application of the largest unit is the best solution economically. Widely separated, non-continuous types of deposits, divided ownership of large deposits, limited markets for a particular product and a host of many related problems may determine, however, that in one type of application smaller sizes should be applied for maximum economy and efficiency.

Two general types of shovel/dragline application are reviewed, in the first instance a 7-yd^3 stripping shovel being compared with a dragline of similar size capacity. For properties where overall economics would warrant, the comparison of a 45-yd^3 stripping shovel and a 35-yd^3 walking dragline is made. These particular sizes were chosen as both units represent approximately the same capital investment—\$2 000 000. In addition, both types of unit have been in use in the U.S.A. in a wide variety of applications for some twenty years and reliable information is available on their cost and production.

Considering first the type of property where smaller equipment would be used, comparison will be made by employing the Marion 183-M—a diesel-powered, two-crawler machine which can be operated both as a dragline and stripping shovel. Two diesel engines are utilized on the 183-M, one engine providing the power for the main hoist and for dragline, and also drag function. The smaller diesel engine drives a generating set which provides the power for d.c. motors, two of which are applied to the swing function while one is used for the crowd function.

For the segment of the industry classified in the ‘smaller-stripper’ class, generally speaking, the two-crawler diesel-type units are preferred. Approximately ten to fifteen years ago in this same industry small diesel walking draglines were applied. Engineering developments and manufacturing processes developed larger two-crawler type units, however, and they have, for all practical purposes, taken over the complete industry. Two-crawler type units are preferred because of their greater ease of mobility and since they can be disassembled and re-erected far more easily and cheaply than similar-size walking draglines.

Figs. 1 and 2 illustrate the application of the 183-M shovel and dragline respectively. (In all the diagrams certain values are assumed, based on a general average of many stripping operations. Individual factors for any one specific operation will obviously vary from those utilized.)

It was assumed that the material in the bank condition would swell 25 per cent in volume when placed into the spoil bank. The slope of the highwall or bank is assumed at three vertical to one horizontal with the slope on the spoil being taken at five horizontal to four vertical. In all instances it was further assumed that the cuts are taken on a normal straight cut and no provisions are made for gain or increase in spoil area that would be available by outside curves or throwing the bucket in a dragline application.

In Fig. 1 the width of cut taken by the shovel is 40 ft with a 20-ft width of bench. It should be noted that a maximum of approximately 30 ft of overburden can be handled without rehandling any spoil or employing any other type of auxiliary equipment. In Fig. 2 the width of cut is 50 ft, no allowance being made for bench widths as this would directly penalize the reach of the dragline. It will be seen that a height of approximately 48 ft of overburden can be handled by the 183-M dragline.

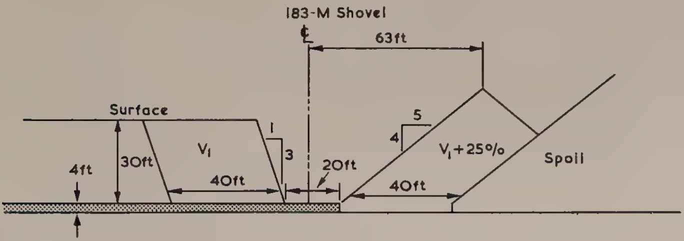


Fig. 1.—183-M Shovel: boom 70 ft; handle 42 ft; dipper 7 yd³.

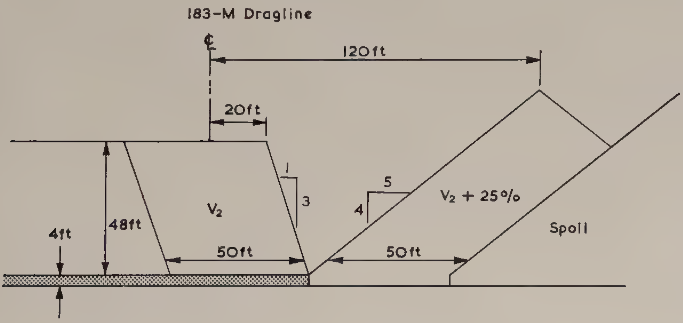


Fig. 2.—183-M Dragline: boom 140 ft; bucket 7 yd³.

For this particular excavator the owners or users utilize many methods to ascertain the working range. As is seen in the sections, neither a great deal of height nor width can be moved by the example

7-yd^3 shovel or dragline. Bulldozers are utilized both to move bank material and to assist in rehandling spoil. In many similar operations the excavator itself rehandles up to 100 per cent of the material. In this type of operation the use of explosives in actually casting or moving the overburden—not only breaking or shattering it—is constantly increasing.

TABLE I.—*Estimate of output*

	183-M Dragline	183-M Shovel
Size of bucket/dipper, yd^3	7	7
Bucket/dipper factor, per cent	65	70
Actual bucket/dipper load, bank yd^3	4·6	4·9
Cycle time, s	60	55
Cycles/h	60	65
Output/digging h, bank yd^3	276	319
Operating efficiency, per cent	85	85
Output/scheduled hour, bank yd^3	235	271

An estimate of the output from a production point of view is given in Table I. For the dipper and bucket factors it should be pointed out that this one value includes the corrections for the swell of the material, inherent loading characteristics of the bucket and dipper itself and such other items as spillover. Specific type of material will have a great deal to do with this factor; for example, comparing a sandy clay to a drilled and blasted hard sandstone. Note that the bucket or dipper load results in terms of bank yards—in all mining applications a definite measurable amount. The cycle times of the units are given as being long-term

TABLE II.—*Estimate of direct operating costs*

	183-M Dragline	183-M Shovel
Operator/h	$4·00	$4·00
Oiler/h	3·00	3·00
Fuel oil at $0·15/gal*	4·50	4·50
Lubricating oil	0·50	0·50
Maintenance (includes all phases)	5·17	6·77
Total/h	$17·17	$18·77
Cost/bank yd^3	$·073	$·069

* U.S. gallon.

averages. Individual cycle times can be considerably lower, but for a true long-term day-to-day, year-to-year operation, the stated values should be applied. Finally, consideration is made for operating efficiency which corrects for minor 'moveups' of the machine, minor stoppages, etc. The overall estimate is conservative and does not represent the peak maximum values obtainable in either type of unit.

To complete the preliminary comparisons of the 183-M shovel and dragline an estimate of the direct operating costs is given in Table II. No breakdown is provided for indirect costs as methods of calculation vary widely, depending on local regulations on taxation and allowable depreciation rates. The approximate capital cost (either 183-M shovel or dragline) is approximately $390 000 ex factory. The costs given are based on current rates in the U.S.A. The values can be factored to applicable values to certain geographic areas to gain an exact figure, but the general relationship between dragline and shovel will not alter. Of great importance in the calculation shown is the fact that the value for maintenance represents *all* items—both labour and materials—such items as cable, weld rod for bucket/dipper and teeth also being included.

Before drawing any conclusions from the brief analysis of the 7-yd^3 shovel/dragline, the same consideration and study will be made on larger equipment, a 45-yd^3 stripping shovel and a 35-yd^3 walking dragline being compared (see Table III). As mentioned earlier, both of these units require an investment of approximately $2 000 000.

TABLE III.—*Comparison of shovel and dragline*

	Shovel	Dragline
Classification no.	5561	7800
Boom length, ft	120	220
Dipper handle length, ft	$78\frac{5}{6}$	
Crowd rack length, ft	$48\frac{1}{2}$	
Capacity dipper/bucket, yd^3	45	35
Maximum digging height/depth, ft	$106\frac{1}{2}$	141
Maximum dumping height, ft	$80\frac{1}{4}$	71
Maximum dumping radius, ft	$120\frac{1}{4}$	211
Total a.c. horsepower of motor generator set(s)	1500	2500
Total d.c. motor horsepower of:		
hoist	850	1000
crowd	188	
drag		1000
propel (from hoist)	850	1000 (from drag)
swing	$412\frac{1}{2}$	550
Working weight, lb	3 350 000	3 435 000
Bearing pressure, lb/in.^{2}	52·4	12·1

The 5561 shovel and the 7800 dragline have been in use for some twenty years and reliable information on the application of these two units is therefore available. It should be pointed out that modifications in boom lengths, ranges and capacities are possible, depending upon the specific application, the specifications given in Table III being the so-called standard ones.

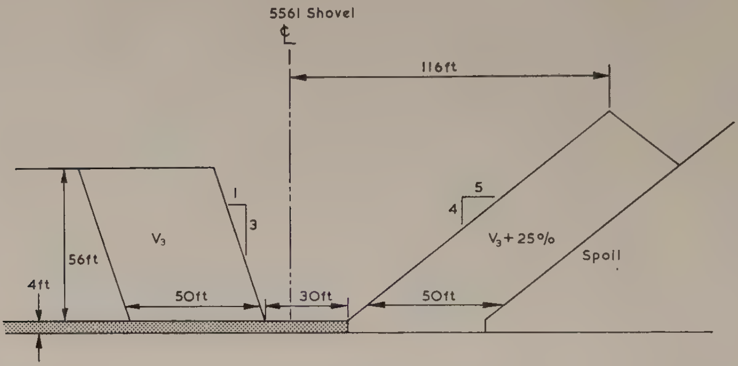


Fig. 3.—5561 Shovel: boom 120 ft; dipper handle 78 ft 10 in.; crowd handle 48 ft 6 in.; dipper 45 yd³.

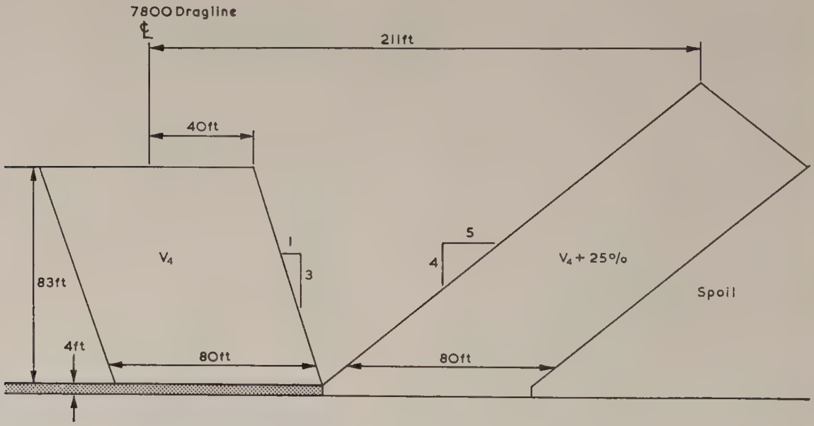


Fig. 4.—7800 Dragline: boom 220 ft; bucket 35 yd³.

Figs. 3 and 4 represent the typical section for the 5561 shovel and the 7800 dragline respectively. For the 5561 shovel a bench width of 30 ft was assumed and a width of cut of 50 ft. No bench was provided for the dragline section and the width of cut is 80 ft.

The maximum height of spoil that can be handled by the 5561 shovel is approximately 56 ft and for the 7800 dragline approximately 83 ft. Some slight variation in the sections can be introduced by varying the pit widths.

As with the 183-M, production or output estimates for both these units are given (Table IV). The various elements considered previously also apply for these estimates: the only fact that has been altered is that for both units the long-term average cycle time will be 60 s/cycle.

TABLE IV.—*Output estimate*

	5561 Shovel	7800 Dragline
Dipper/bucket capacity, yd^3	45	35
Dipper/bucket factor, per cent	70	65
Dipper/bucket load, bank yd^3	31·5	22·8
Cycle time, s	60	60
Cycles/h	60	60
Output per digging hour, bank yd^3	1890	1370
Operating efficiency, per cent	85	85
Output per scheduled hour, bank yd^3	1607	1166

Table V provides a comparison of the direct operating costs currently prevailing in the U.S.A. As both of the units are Ward-Leonard Electric type excavators the cost of electricity is a significant item. An average value was taken since great variations exist on kWh costs. For estimating indirect costs or depreciation, the value of the units has already been provided.

TABLE V.—*Estimate of direct operating costs*

	5561 Shovel	7800 Dragline
Operator/h	\$4·00	\$4·00
Oiler/h	3·00	3·00
Groundman/h	2·50	2·50
kWh cost based on \$·015/kWh	9·64	10·49
Maintenance (includes all phases of maintenance)	38·57	25·65
Total	\$57·71	\$45·64
Cost/yd^3	\$0·036	\$0·039

DRAGLINE

The greater reach and dumping radius of the dragline enable deeper overburden to be removed. For the same capital outlay considerably more depth of overburden can be stripped, thereby making available those mineral deposits which would be uneconomic for a shovel operation.

The dragline is the more versatile and easily manoeuvred unit: it is located on or near the ground surface, which immediately eliminates certain problems; slides in either the bank or spoil cannot block or impair its operation; water presents little difficulty as the runoff is easily controlled; and seepage is a small problem on the surface when compared to being located within the confines of a pit.

Variations in the configuration of the top of the mineral deposit can easily be adjusted for by the dragline and it is of but slight importance if the mineral deposit has localized pitches and rolls since overall planning of the operation can correct for these variables and their effect on required cut widths or spoil area.

If necessary, the dragline can 'chop down' a certain percentage of the overburden. While this is not as efficient as the standard operating procedure of a dragline, it can be used to advantage. Variations in ground topography are eliminated by 'chopping down' and the dragline then guarantees itself a constant level horizon from which to work. An increase in dumping radius can also be obtained by this method, but it must again be borne in mind that this type of operation is not the most economical.

To open a new property it may be necessary to begin with a box-cut, in which case the dragline's reach and location can be utilized very efficiently and easily. When the final cut or last phase of a stripping operation is achieved the dragline can take advantage of spoiling both into the spoil area and also on top of ground surface, thereby being able to expose the mineral deposit to a greater depth.

Rehandling of spoil is also a simple operation although, of course, adding to the overall cost. Conditions and overall economics may, however, warrant a certain percentage of rehandling and were it not for the fact of being able to rehandle economically several well known operations would not now be in existence. The respective mineral deposits might well be uneconomic to recover by other means and would therefore be lost.

If different conditions are encountered, the boom of the dragline can be lengthened or shortened fairly easily. To a certain extent, working ranges can be varied by altering the boom angle—simply by raising or lowering the boom. Naturally, bucket capacities would also change in accordance with the allowable loads at varying radii.

SHOVEL

Greater capacity is obtained with a shovel than with a dragline—all the more so in the larger units. For basically the same outlay the 5561 shovel with its 45-yd^3 dipper provides approximately 30 per cent more capacity than the similarly priced 7800 dragline.

Besides the greater capacity inherent differences in application and operation permit greater production. The dipper on a shovel is fastened by means of a solid connection to the driving mechanism, total and positive control being possible always. A dipper on a shovel therefore generally tends to have a higher loading factor than a dragline bucket of same size. Scooping up through the boom will obviously assist in loading the dipper when compared to dragging a bucket by cable through the same material. As the material becomes harder or rockier, the greater will be the advantages in loading for the shovel. Considerably less expense will be involved for drilling and blasting in the shovel operation. Blockier material will have less overall effect on the shovel's ability to load material efficiently.

In the application of the shovel, a great percentage of the material is loaded in an area directly in front of the unit and, at the same time, the material is dumped or spoiled within a relatively short radius of the loading point. The result is that the cycle time will generally be lower for a shovel than for a dragline. The difference becomes greater or more noticeable in those dragline applications where long booms are utilized, when considerable 'chopping down' is performed, or in deep overburden.

Although being located near the original ground surface can be an advantage for the dragline, there are disadvantages too. The location of the shovel on top of the mineral deposit is not controlled by original topography. Thus the possibly inefficient and expensive preparation of a working bench is eliminated, particularly in extremely undulating areas.

The shovel usually maintains a certain berm width in the pit, which makes a roadway available for haulage units. Where haulage cannot be done on the beds below the mineral (for example fire clay below coal) this berm is a definite advantage.

It can be concluded that with the shovel the cost per cubic yard of overburden removed is lower than with the dragline.

CONCLUSIONS

It must be remembered that the existing conditions determine the selection of dragline or shovel. No two properties are exactly alike and information or experience gained in one particular operation cannot therefore be simply applied to another. Variations in conditions may be minute and seemingly insignificant, but they may determine the difference between a successful and highly efficient operation and one that is constantly beset with problems and inefficiencies. It is hoped that the present paper will provide a general guide to the uses of dragline and/or shovel. It is stressed, however, that it is a guide—not a final solution.

Acknowledgements.—The writer wishes to acknowledge the assistance and co-operation of certain members of Marion Power Shovel Co.—the late Mr. L. C. Morley, Mr. G. Boulter and Mr. J. Weis in particular.

621.879.29:622.271.3(470) *Paper 26*

Use of Walking Draglines for Intensive Opencast Coal and Ore Mining

N. V. MEL′NIKOV, Academician, Professor, Doctor of Science

N. N. MEL′NIKOV, Mining Engineer
Skochinsky Mining Institute, U.S.S.R.

SYNOPSIS

A wide range of walking excavators is being developed in the Soviet Union. Technical details of various models are given in the paper, followed by descriptions of the application of these draglines to digging drainage cuts, driving haulage cuts, erecting spoil banks, stockpiling, overburden removal, loading into dump cars and, finally, mining bedded deposits.

In the U.S.S.R. walking draglines are widely used in opencast mining and construction work for extracting and loading rocks, coal, iron ore and other useful minerals into dump cars and bunkers, as well as for stockpiling material.

Technical data of draglines made in the U.S.S.R. are presented in Table I.

TABLE I.—*Dragline data*

			Models			
Specifications	4/40	6/60	14/75	15/90	25/100	50/125* 80/100
1	2	3	4	5	6	7
Bucket capacity, m^3	4	6–10	$\frac{14}{20}$	15	25	50–80
(yd^3)	(5·2)	(7·8–13)	$\left(\frac{19\cdot3}{26\cdot2}\right)$	(26·2)	(32·7)	(65·4–104·6)
Boom length, m	40	60	$\frac{75}{65}$	90	100	125–100
Boom angle, degrees	25–35	30	20	30	30	30
Max. dumping height, m	13·4–19·4	24	20	42	47	58
Max. dumping radius, m	39–35	57	67	83	95	121
(ft)	(127·9–114·8)	(187)	(219·8)	(272·3)	(311·6)	(396·9)
Max. digging radius without bucket throw, m	45	58	77·5	85	95	125
Max. digging depth, m	32–26	35	40	41	47	58
Platform revolutions per min	2·01	1·3	1·43	1·25	1·28	—
Walking speed, km/h	0·47	0·2	0·06	0·06	0·06	0·06
Average unit pressure on the ground, kg/cm^2						
in operation	0·43	0·735	1·0	0·8	1·0	1·0
in walking	1·05	1·05	1·5	1·27	1·5	—
Cycle time at 110° swing, sec	55	60	65	68	70	70
Working weight, t	186	520	1400	1450	2500	5100

* Dragline 50/125 is to be mass-produced.

There are several original features in the dragline designs: cable-stayed booms for the larger models, a hydraulic walking mechanism, light-weight buckets of a new type with a continuous cutting lip, etc., special steels being used for many important items.

EXAMPLES FROM THE KUSTANAI REGION

The use of draglines for a variety of purposes is well illustrated in the exploitation of ore deposits in the Kustanai region—where the overburden is 100 m deep and consists of soft strata with a high water content. The total water inflow in the open-pits amounts to 3500 m^3/h. Medium-hard iron ore is enclosed in hard rocks.

Digging Development Cuts for Drainage of Deposits

During the construction of the quarries at these deposits a preliminary layer-by-layer drainage of overburden strata was performed by the development cuts dug by draglines. This technique allowed the development work to continue instead of having to wait until the deposits were drained with the aid of normal methods such as underground workings through filters and deep drainage holes drilled along the boundaries of the quarry. Thanks to the dragline technique the deposits were put into commission two or three years earlier than scheduled.

The essence of this method is that a dragline (4/40, 6/60) cuts a drainage cut located in accordance with the mine plan to the maximum digging depth, parallel to the axis of the future entrance or working cut. The rock is piled up and mechanical shovels then load it into dump trucks which remove the rock to the spoil dumps outside the deposit.

Centrifugal pumps are installed on floating metal pontoons in drainage cuts and water is pumped out. This ensures that the water-level is lowered to the bottom of the cut and rocks in the adjacent area are drained. As a result, satisfactory conditions are created for mechanical shovels to operate at the drained level.

As the rock is extracted a new drainage cut is excavated to a lower level, water is drained by floating pumps and the water-table is again lowered. Both small and large areas are drained with the help of these methods.

Driving Haulage Cuts

Driving of haulage cuts by mechanical shovels is difficult owing to the fact that the loading has to be done in a dead end, often under wet conditions and with excavators working on an incline. Under these conditions, however, draglines can effect the driving of the upper part of deep cuts without the rock having to be handled by road or rail transport before stocking at the side of the cut.

This use of draglines considerably reduces the cost of the work and increases the rate of driving.

The main haulage cut at the Sokolovo quarry was driven in this way by a dragline (model 14/75) and wet loose rock from the cut was moved by the dragline on to the side of the cut requiring two and three rehandlings.

The method was especially effective during the construction of the railway cutting, designed for hauling from the top of the orebody.

The particular advantages were:

(*a*) compared with the ordinary method of driving cuts with face shovels the number of dump trucks required was reduced since the dragline lifts rock to a great height and the trucks are not required for this operation;

(*b*) dragline 14/75 ensures intensive lowering of water-table in the area where a cut is being driven;

(*c*) a number of working places are made available for mechanical shovels which can load rock into dump trucks on two levels at the same time; and

(*d*) the time required for drain cut construction is reduced considerably with resulting economies.

The construction of this opening cut, the total volume of which reached 4 500 000 m³, was accomplished within two years with a great saving in cost.

Forming of Spoil Banks

Construction of spoil dumps with the aid of mechanical shovels by gradually dumping rock is a long and labour-consuming process. However, it is always necessary to raise the dump quickly and as high as possible to form a side tip for rail wagons or dump trucks. This provides for high receiving capacity and creates conditions for intensifying the removal of overburden.

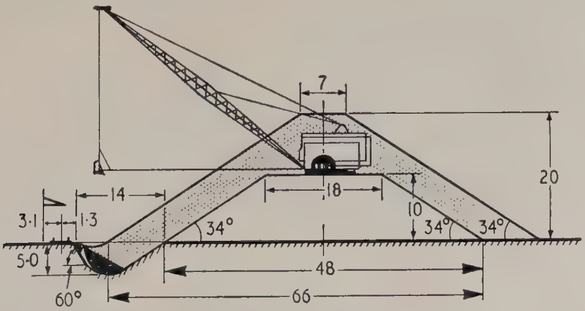


Fig. 1.—Use of dragline 4/40 at Sokolovo quarry to build central stockpile embankment. (Height of embankment, 20 m.)

At the Sokolovo and Sarbai quarries the spoil dumps were erected by draglines, model 4/40, up to a height of 20 m and with the total length of tip side amounting to 4·75 km (the working part being 2–2·5 km long). At the Sarbai quarry a dragline, type 4/40, was stationed on the ground surface and dug rock from the surface, erecting a stockpile. On average the rate of forming the tip bank reached 150–200 m per month. At the Sokolovo quarry the stockpiles were erected in a different way (Fig. 1). The rock intended for the spoil dump was transported by rail from the quarry, unloaded in the dragline excavation ditch and then lifted on to the stockpile up to the designed height of 20 m. In this case the rate of erecting the spoil bank was slower, equalling 40–60 m per month.

Stockpiling by Means of Draglines

In some open-pits stock banks are erected by stackers or mechanical shovels. For some time past draglines were also used for this purpose. The changeover to the building of stockpiles by dragline models 4/40 and 6/60 made it possible:

(1) to increase the distance slewed for each step of the railroad track and decrease the volume of the track-laying work 3 to 5 times. It

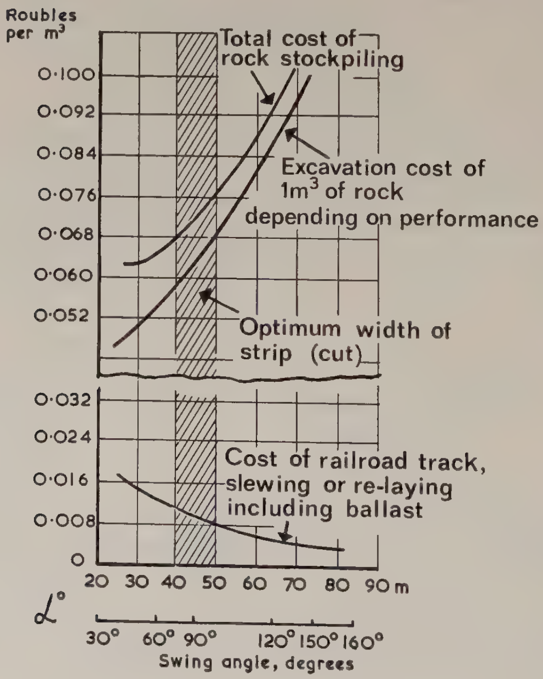


Fig. 2.—Cost curves for building stockpiles or spoil heaps by dragline 4/40

(1 rouble = approx. 8s.)

has become possible to lay the railroad track on ballast owing to the fact that the rails remain in the same place for a long period, thus making possible an increase in the speed of trains;

(2) to reduce the work of the railroad transport in the lifting of rock to a higher bank level since draglines recast rock above the level of the railroad; and

(3) to create more reliable conditions for laying rails on dumped ground. (If, for example, the tip bank length is 2 km the excavation cut is complete in $1\frac{1}{2}$ to 2 years, so giving enough time for the ground to consolidate.)

When erecting a stockpile it is best to keep to an angle of swing of the dragline under 90 to 110° to ensure the best performance of the draglines and a low cost per metre of stockpiling (Fig. 2). Experience has shown that slips occur less frequently in bank faces erected by dragline than by mechanical shovel. The recommended method of erecting a stockpile by dragline is illustrated in Fig. 3.

Trains approach the dragline by two tracks, so increasing dragline efficiency which after a time reaches 90 per cent. When two railroad tracks are available a dragline recasts rock on to the stockpile operating at angles ranging from 0 to 90°, thus giving maximum performance.

This method provides for high reception capacity of the stockpile. Each railroad track is used twice (for two cuts). Where a second stage is formed, the capacity of the stockpile increases 1·5 times. Railroad tracks are laid on the ground surface and not on made-up ground. This facilitates their utilization and reduces expenditure on current maintenance of the railroad.

Ditches running along the stockpiles increase the stability of the slopes.

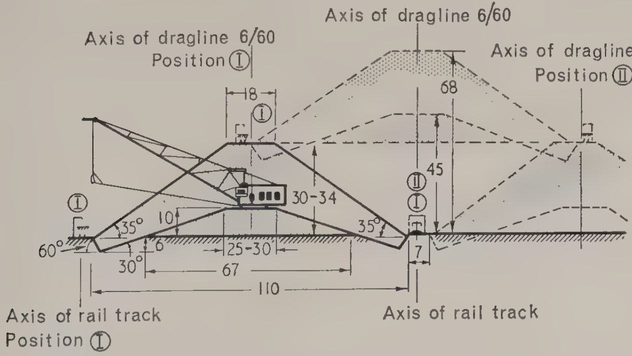


Fig. 3.—New method of building stockpiles or spoil dumps by using dragline excavators.

Re-laying of railroad tracks on stockpiles decreases by a factor of from 5 to 8 compared with stockpiles erected by mechanical shovels and by from 50 to 80 as compared with those erected by plough excavators. Reception capacity of one running metre of stockpile increases 8 times as compared with fills made by mechanical shovels.

The cost of 1 m^3 of stockpile formed by dragline is about 60–75 per cent of the cost of one formed by face shovels.

Overburden Removal by Draglines—Model 14/75

Draglines of type 14/75 are used in excavating overburden (quartz–glauconite sand) amounting to 30 m in thickness and occurring over the ore at the Sokolovo and Sarbai quarries.

At the beginning of the overburden removal, until the water-table level was sufficiently lowered, the sands were wet. Under these conditions the draglines stripped the ore. At present the same excavators (14/75) removed the sand, piling it up to a height of 140 m. From these heaps the sand is rehandled into dump cars by mechanical shovels (type ЭКГ–4) and is removed to the external dumps by rail.

The average monthly output per dragline type 14/75 is 250 000 m^3.

Under these conditions the use of this type of dragline for removing overburden is determined by the following factors.

(*a*) The necessity of cutting three transportation levels under confined conditions and in wet rock is avoided. The volume of work on cutting out the edges, for example at the Sokolovo quarry, has been reduced by 6 000 000 m^3.

(*b*) Railway transport is relieved of some burden as rock is lifted by the dragline to a height of 30 m, equivalent to the transportation of rock by rail to a distance of 1·5 km.

(*c*) Thanks to the drainage of sand the railroad track is laid on dry foundation.

(*d*) The cost of excavating one m^3 of rock made by the model 14/75 dragline has been reduced by 13–25 per cent compared with the face shovel and rail haulage method.

Loading Rock into Dump Cars by Dragline Model 6/60

For the purpose of draining overburden rocks at the Sarbai quarry working trenches are driven along the boundary of the area to be worked each year by means of draglines of type 6/60.

Rock is loaded by draglines directly into dump cars, the load-carrying capacity of which is 80–90 tons. Since in the process of unloading the dragline bucket some material is dropped, to prevent the railroad tracks from being littered a bulldozer drives a ditch along the railroad track on the side opposite to the dragline to receive the debris from the dragline bucket.

The dragline is positioned at the same level as the railroad track and unloads rock at an angle of turn of the boom 90° to the track. The railroad overhead electric conductors are moved to a distance 5–7 m from the side of the track. The length of the conductor section moved is about 20 m. When loading of the dump cars is completed, the train is driven to the dead end. The electric locomotive then gathers momentum and passes the de-energized section by inertia with the pantograph dropped.

The dragline output has reached 103 000–110 000 m^3 per month.

Experience has shown that rock may be loaded by a model 6/60 dragline directly into dump cars in the following cases:

(1) when driving entrance, working and draining cuts in wet ground where the use of mechanical shovels is difficult;

(2) when loading rock into railway cars using upward and downward digging in an end face—useful when cutting a new level from the flanks, for which a dragline is positioned on an intermediate level and loads rock at 90° into railway cars located above and below the level of the dragline; and

(3) when loading rock by using upward and downward digging in a main face.

It is expedient to use this layout while excavating rock at the flanks, i.e. in the sides of the quarry, when the contours of the quarry approximate those designed and there is no necessity to maintain a greater number of transportation levels.

A dragline is positioned on an intermediate level and loads rock at an angle of turn ranging from 0 to 180°.

Actual figures show that in comparison with the ordinary method of excavation work done by mechanical shovels the cost of one m^3 of rock loaded by a model 6/60 dragline into dump cars is higher by 5 to 10 per cent but, taking into account the simplicity of the method and the number of advantages, the loading of rock by the dragline directly into dump cars is considered to be preferable. On the whole in the exploitation of the Sokolovo and Sarbai iron ore deposits the volume of mine work done effectively by draglines constitutes 30 per cent.

USE OF HEAVY DRAGLINES FOR MINING BEDS

In the Soviet Union draglines are widely used for mining bedded deposits—coal, iron ore, manganese ore and other useful minerals.

The normal methods are strip mining or cut-and-fill, when overburden material is handled by draglines on to spoil banks situated in mined-out areas. In case there is not sufficient room to place the excavated rock in the spoil dumps, rehandling is used. For this purpose draglines are stationed on the spoil banks.

To load out the coal or ore, mechanical shovels with road or railway transport are employed. Roads are laid on the bottom of a bed. In such methods one, two or more loading benches are excavated.

If the thickness of overburden strata is very great, and beds occur with the dip not exceeding 10°, combined methods of mining are applied.

As in the Urals at the Vakhrushev coal pits for mining a coal bed 30 m thick, the rock overburden of upper benches is excavated and transported to spoil dumps by road or railroad, while the lower bench is excavated by the cut-and-fill method. In this case rock is cast on to spoil banks situated in a mined-out space.

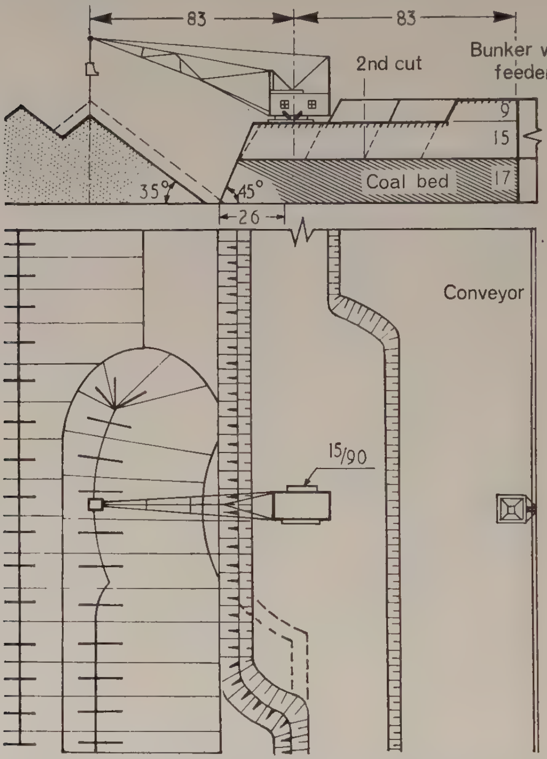


Fig. 4.—'Transportless' method of mining using the cut-and-fill system with a single walking dragline for both stripping and loading.

In connection with the wide use of draglines, one method which has been evolved, called 'excavator quarry', uses one and the same dragline stationed on an intermediate overburden level (Fig. 4) to remove covering rock and load the mineral. There is no need to drive ordinary entrance and working cuts for mining these deposits since the useful mineral is transported along the ground surface.

From the same position a dragline first removes overburden, recasting it from the upper and lower parts of the bench to the spoil dump. Then, after the bed is stripped, the dragline excavates coal, loading it into a self-propelled bunker or stockpiling it on the surface. From the bunker or stockpile an excavator reloads coal or ore on to a conveyor, dump trucks or cars. Experience shows that draglines with buckets of 15 m^3 or more are capable of excavating hard rock or ore if first prepared for excavation by blasting. Wear of buckets and ropes is not unreasonable.

After mining out coal or ore the dragline excavates overburden from the next cut. Thus the dragline alternates removal of overburden and mining of mineral. If two or three draglines are employed in an opencut, mining goes on continuously round the clock.

The ‘excavator quarry’ method is at present used in mining brown coal and iron ore in the Tula region. Also, the Ackerman deposits of iron ore in the Orenburg region are in the process of preparation for mining, and a plan has been drawn up for opencasting of coal fields in Nasarovo, Siberia, using the ‘excavator quarry’ method. The Nasarovo coal bed reaches 16 m in depth and is to be excavated by draglines of type 50/125 (bucket capacity of 50 m^3). Estimates show that in these coal fields the cheapest coal will be obtained with an output of 80 t/man-shift.

In the U.S.S.R. great importance is attached to opencast mining. At present this method is used for mining 22 per cent of coal, 55 per cent of iron ore, 50 per cent of ore of non-ferrous metals, 40 per cent of manganese ore, and almost 100 per cent of non-metallic minerals.

Within the next ten years it is planned to increase the opencast output up to three-quarters of the total volume of the output of useful minerals. In particular, 500 000 000 tons of coal a year by opencast mining methods only is aimed at. Draglines will therefore play a large part in this campaign and will make up not less than 30 per cent of the total fleet of excavators.

622.015.36:622.341.1(425) *Paper 27*

Restoration of Ironstone Workings in the Midlands of Great Britain

T. W. JONES, F.R.I.C.S., Certificated Mining Surveyor
Northern Area Chief Surveyor, Stewarts and Lloyds Minerals, Ltd., Melton Mowbray, Great Britain

---

SYNOPSIS

Ironstone has been worked in the Midlands with varying degrees of intensity from the middle of the nineteenth century and, as far as possible, the land worked has been restored to agriculture. During the past 30 years modern machinery and techniques have assisted in the problem of dealing with overburden, but at the same time have brought various new difficulties.

The great improvements in earthmoving equipment have made it possible to restore worked land to agriculture as a general policy. The Minister of Housing and Local Government—by virtue of the Town and Country Planning Act, 1947, and subsequent legislation—has the overriding authority as to the degree of restoration to be carried out. The financial provisions contained in the Mineral Workings Act, 1951, have been of great help in carrying out the works. Approximately 15 500 acres have been worked for ironstone in the Midlands, including 1500 acres at present being used for quarry operations. Approximately 62 per cent of the land has been restored to agricultural use and a further 15 per cent afforested, while 8 per cent has been used for other purposes, such as industrial development.

Ample evidence is available that ironstone was worked in Great Britain during the Roman occupation and in mediaeval times up to the sixteenth century, particularly in the Weald and in the East Midlands. The introduction in the sixteenth century of rigorous timber laws cut off the source of fuel and caused the production of iron to cease in the latter area for more than 200 years. Interest in the East Midlands as an ironstone-producing area was revived in the mid-nineteenth century, the big railway construction programme playing an important part in the revival. Ironstone exposed in many of the railway cuttings drew attention to the possible importance of the bed and subsequent technical advances in steel production led to renewed exploitation. Output continued to rise, replacing the decline in production of Cleveland ores, and approximately two-thirds of the present-day home iron ore production is obtained from the South Lincolnshire, Leicestershire, Rutland and Northamptonshire district of the East Midlands.

Early working of the ironstone field was manual and therefore concerned only ironstone outcrops or ironstone under very shallow cover.

Uncovering of the stone entailed hand loading the spoil into barrows and the use of a 'plank and trestle' bridge spanning the ore wagon track, depositing it evenly on the waste side in preparation for its return to agricultural use. The ore itself was also hand loaded into rail wagons (Fig. 1).*

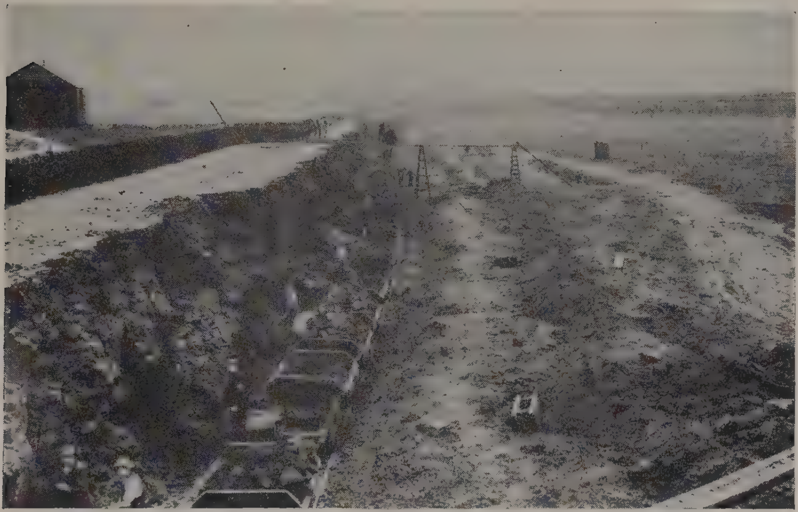


Fig. 1.—Early method of ironstone extraction. The ironstone is being hand loaded into rail wagons. In the background the trestle bridge provides the means of transferring hand-bared material across to the spoil side for restoration to agriculture.

Increased demand for ironstone, coupled with the realization of the importance of the field, necessitated the exploitation of ironstone under deeper cover, and in 1897 the first steam-navvy operating in conjunction with a transporter was introduced for stripping overburden. A similar type of machine had been introduced twelve months previously to replace hand loading the ironstone. Subsequently several short-jibbed steam shovels were developed, but in use a transporter was required to carry the spoil for depositing on the waste side. In 1899 the forerunner of modern stripping navvies was introduced in the form of the 'Wilson' shovel, designed with a long jib to remove and dump overburden without the use of a transporter.

The steam shovel was not considered suitable for baring shallow overburden, but it was not until 1916 that a dragline was used to replace hand labour. As the need to exploit ironstone at greater depths increased so too did the size of machines. At present the largest dragline in operation in this orefield will deal with overburden up to 100 ft in thickness; it has a boom length of 303 ft, a bucket capacity of 22 yd^3 and weighs 1775 tons. The capacity of the bucket of the largest shovel is 15 yd^3. Developments have also occurred in the design and manufacture of subsidiary machinery to assist in the baring of areas of deep overburden and in the subsequent restoration of the worked land, bulldozer and scraper units proving invaluable.

---

* All photographs appearing in the paper are reproduced by kind permission of Stewarts and Lloyds Minerals, Ltd.

HISTORY OF RESTORATION

Before 1933 practically all ironstone workings were being carried out under conditions of shallow overburden and as a consequence no difficult restoration problems arose. Iron ore is privately owned and working rights have been acquired by negotiating leases with the mineral owner. The earlier leases were generally in line with the shallow conditions, some being 'full restoring' leases—that is leases where the lessee has no option but to restore—and others putting the emphasis on complete restoration, with a clause covering the possibility of failure to comply almost as an afterthought. With the deepening of the overburden, and as new leases were negotiated or old ones renewed, there was a trend to recognize that full restoration would no longer be an economic proposition—reflected in a 'restoration' clause whereby a definite payment of compensation per acre could be made to the lessor in lieu of restoration. The amount of compensation varied according to the lease, being a matter of negotiation, and ranged from £20 to £60 per acre. The unrestored land, of course, remained the property of the lessor to do with as he saw fit. As workings in the deeper cover expanded so too did the amount of worked land left unrestored because of the more difficult conditions and the non-availability of suitable restoration machinery. Concern was felt at this so-called devastation of the country and in 1938 a report was called for by the Government on the matter and the steps being taken towards the restoration of worked ironstone land. For this purpose a committee, headed by the late Lord Kennet, was appointed which advocated the setting up of an 'Ironstone Areas Restoration Board', made up of representatives of the County Councils, the Forestry Commission, the companies and the ironstone owners. It was proposed that the Board take the question of restoration out of the hands of the companies and owners and be empowered, by means of a levy on the mineral owners, the companies and the local authorities, to raise an income of £40 per acre of land it took over. Nothing transpired from these proposals, possibly because of the commencement of the Second World War, but in 1944 Mr. A. H. Waters was appointed to make a further report—which included proposals put forward by the companies and owners also suggesting a controlling body, but unlike the Kennet report the representatives were to be drawn from central Government departments instead of from the local authorities. Furthermore, it was proposed to relieve the latter from any levies. The report also reached the conclusion that restoration should run parallel with extraction.

The Town and Country Planning Act, 1947, followed, and brought in stricter measures of control over development of land and buildings. It conferred on local planning authorities power to control opencast and underground mining operations and to allocate iron-ore-bearing lands for future needs. Applications for permission to develop properties were required to be submitted to the local planning authorities, usually the County Councils concerned. In the case of all applications for the winning and working of iron ore in Leicestershire, Northamptonshire, Oxfordshire and Rutland, and in certain parts of Lincolnshire and Warwickshire, the Minister of Housing and Local Government has directed that they shall be referred to him for decision instead of being dealt with by the local planning authority.

Since development was taking place in many of the ironstone application areas, as an interim measure under the 1947 Act a special order, 'The Ironstone Areas Special Development Order (S.I. 1950, No. 1177)', was brought out. It provided for the levelling of all current workings after 1st April 1951, and for the stripping and replacement of topsoil on a working face having a depth above ironstone of up to 35 ft for nine-tenths or more of the length of the face or within 400 yd of the end (not being a tied end). Provision was contained in the Order for the exemption of land intended after working for the deposit of waste material or for land where the overburden to be removed contained a high proportion of hard and unfriable limestone.

In issuing his decision on an application the Minister has laid down the degree of restoration to be carried out and it is the duty of the planning authority to satisfy itself that the required standard has been reached or to issue an enforcement notice to cause such a standard to be reached.

In general it is laid down that from the date of the Minister's decision letter all the land within the area covered by the permission from which overburden is removed in the course of mining operations shall, after working, be restored to such levels and configuration as will facilitate natural drainage and permit the resumption of normal agricultural operations, including the free movement of agricultural machinery on the surface of the land. The topsoiling condition is usually governed by the geological structure of the application area. In areas of shallow cover an operator is required to strip off and replace the topsoil over the whole area, but in deep areas this is only necessary within a distance of up to 500 yd of the end of the pit. The remaining portion is free from such a condition and consequently when levelled it is devoid of topsoil. In areas where the overburden changes fairly rapidly from shallow to deep a combination of the two conditions is imposed. Complete stripping and replacement of topsoil is required on the shallower area to a line where the overburden may be classified as becoming deep, usually at the estimated 35-ft depth line. From this line onwards the conditions pertaining to deep overburden are imposed, the line being shown on the definitive map accompanying the decision. In early decisions time limits for carrying out the levelling conditions were imposed, but experience has shown that better and more economical results have been obtained by delaying restoration operations until worked areas of sufficient size are available.

In 1951 the Mineral Workings Act was brought into operation. It established a fund, controlled by the Minister of Housing and Local Government, to provide money for the restoration of land worked for iron ore by opencast methods within a defined Ironstone District. To create the fund the Act provided for the payment of 3d. per statute ton in respect of all ironstone extracted by opencast methods, 2¼d. being contributed by the mineral operator and ¾d. by the Exchequer. The operator is enabled to recoup half his contribution in respect of leased minerals from royalty payments under that lease. Exemptions have been granted in respect of certain lands held in charitable trusts, in which event the contribution by the operator is 1⅛d. per ton only. The Act also empowered the Minister to determine a rate up to which the operator is required to carry out restoration at his own expense, now determined at £110 per acre.

Payments out of the fund can be made as follows:

(1) to iron ore producers in respect of the excess over £110 per acre of the cost of carrying out the restoration conditions of the planning consent;

(2) to local authorities where they carry out any works for levelling worked iron ore land: these are usually areas left derelict by past workings;

(3) to owners of restored lands where they incur expense in the replacement of fixed equipment, such as fences, hedges, drainage, water supplies, buildings, etc.;

(4) to occupiers where the cost of farming exceeds the normal cost on unworked lands; and

(5) to the owners of the land when afforestation is to be undertaken.

RESTORATION

The method of uncovering the ironstone and the subsequent restoration depend upon two main factors: (*a*) the amount of the overburden to be removed and (*b*) the nature of that overburden. The conditions to be met in the East Midlands field can be shown by a brief description of the geology of the area.

Two iron ores are mined—Marlstone and Northampton Sand—both in the Jurassic system. The lower bed, Marlstone, rests on the Middle Lias Clay measures and is itself overlain by Upper Lias Clays, which can persist to depths of 200 ft. Only quarrying operations at outcrops and areas under comparatively shallow cover have, however, taken place.

On top of the Upper Lias Clays lies the the Northampton Sand bed,

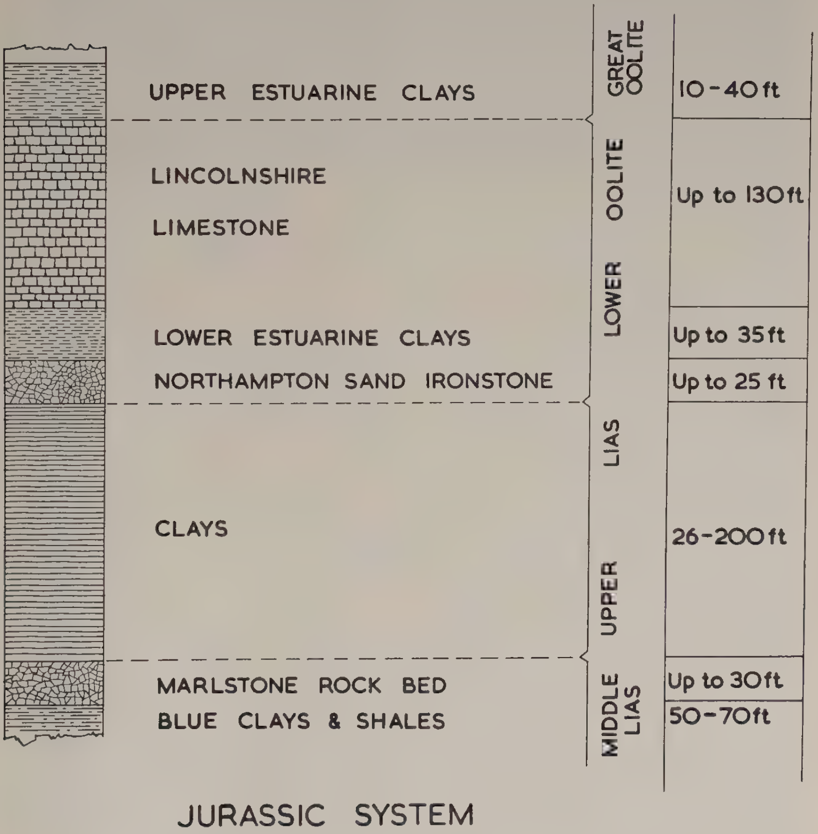


Fig. 2.—Section of Jurassic Series affecting Midlands area.

workings in which have been developed extensively under varied depths of overburden to over 100 ft. Except in areas near the outcrop the bed is immediately overlain by the Lower Estuarine Clays followed by Lincolnshire Limestone. Blasting of the latter bed is carried out to enable the baring machines to dispose of the burden. Glacial action has left behind a deposit of Boulder Clay over much of the deeper areas (Fig. 2).

The uncovering of the ironstone bed ready for extraction follows the same principle irrespective of the depth of the overburden. Movement of both the baring face and rock extraction through the quarry area is made by consecutive strips running the length of the quarry. With the exception of the opening cut of gullet, when the initial topsoil is stacked at one side and the initial substrata are deposited to the rear of the cut, after extraction of the bared ironstone, each succeeding cut is deposited in that previously occupied by the stone. The thickness of the overburden, however, has some effect on the method used for levelling off the deposited substrata and for replacing topsoil.

Shallow Overburden

In these conditions the dragline is an ideal machine, carrying out the topsoil stripping, the baring and restoration and also the loading out of the ironstone. The machine works along the face, operating a cut about 45 ft in width for a length of half-chain at a time. The topsoil is removed with care by the dragline bucket and transferred on to previously levelled spoil on the worked side of the face. The operator, by casting his bucket beyond the boomhead, can get a reasonable spread of the material—any irregularities being evened out by hand labour. The subsoil is dealt with similarly, being deposited in the area from which the ironstone has been removed. As before, the operator's deft manipulation of his bucket leaves the spoil in a level condition, final touches again being supplied by hand labour (Fig. 3).

It is obvious that in this operation little time elapsed between the upheaval of the virgin land and its subsequent restoration. In practice the worked land returns almost immediately to agriculture. It is unusual for the restoration cost to be heavy in these small-scale operations and the total rarely exceeds the standard rate of £110 per acre fixed by the Minister.

It is realized that only a part of a machine's baring time is spent in restoration work, since the rock must be uncovered in any event and the spoil dumped on the waste side. To arrive at a percentage attributable to restoration extensive tests have been carried out with variously sized machines operating in varied depths. The tests took the form of operating a machine over a measured length, during which it was timed to complete the operation of baring—no effort being made to complete restoration. Subsequently, it was timed to give a levelled finish under

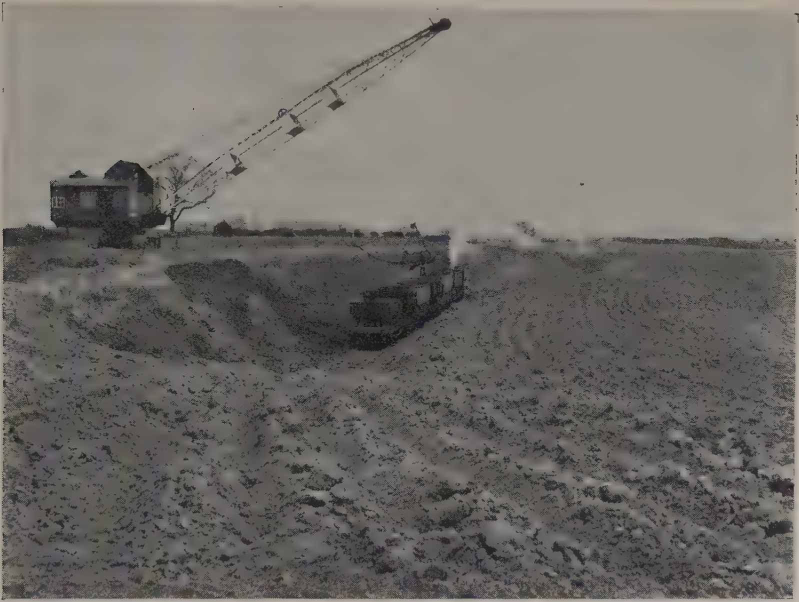


Fig. 3.—Dragline loading iron ore into wagons prior to recommencing the baring of a further strip of land. Baring has been immediately restored and cultivations have been carried out to the limit of the workings.

the same conditions for the same volume of material, including the stripping and replacing of topsoil. The latter, obviously, was a longer operation and the difference in time was converted to a percentage and attributed to restoration. From the results of the many tests graphs have been produced which indicate, reasonably accurately, the percentage of the baring operating cost chargeable to restoration for any machine working under similar conditions at a known depth of overburden.

In general the shallower the overburden the greater is the percentage of the machine's baring time that is directly concerned with restoration, due mainly to the stripping off and respreading of the topsoil. This system, of course, operates only in cases where direct restoration is carried out.

Inevitably, even this simple operation presents problems. Occasions arise when the quality of the ironstone is found to be unsuitable for blast-furnace use and then the inferior stone is either turned over into the waste side with normal baring or it may be left *in situ*. The increase in spoil material is noticeable in shallow conditions and its affect on levelling the worked land to a configuration that will facilitate natural drainage can be considerable. Additional regrading work adds greatly to the restoration costs.

Cover of Medium Depth

Although dragline usage is ideal for handling cover up to about 35 ft in thickness, provided that heavy limestone is not present, direct restoration up to these depths, as carried out by smaller machines in the shallow conditions, has been found too slow and expensive an operation.

Topsoil is stripped off as a separate operation, allowing the dragline to excavate the substrata and deposit them in a fairly rough condition on the waste side. Levelling is completed by bulldozing units before the spreading of the topsoil.

The stripping of the topsoil is carried out according to the prevailing circumstances. In the early working of an area it is probable that there may be no levelled spoil to receive the topsoil and its storage at a convenient place is thus necessary. Such storage is carried out (*a*) by the baring machine itself taking off the topsoil and placing it in windrows clear of the baring line; (*b*) by the baring machine loading the topsoil direct into a lorry, dumper, etc., by which it is carried away and stored at a convenient place; or (*c*) by a scraper unit stripping and carrying it away for storage.

Subsequently, the topsoil is reloaded in vehicles for transporting to;

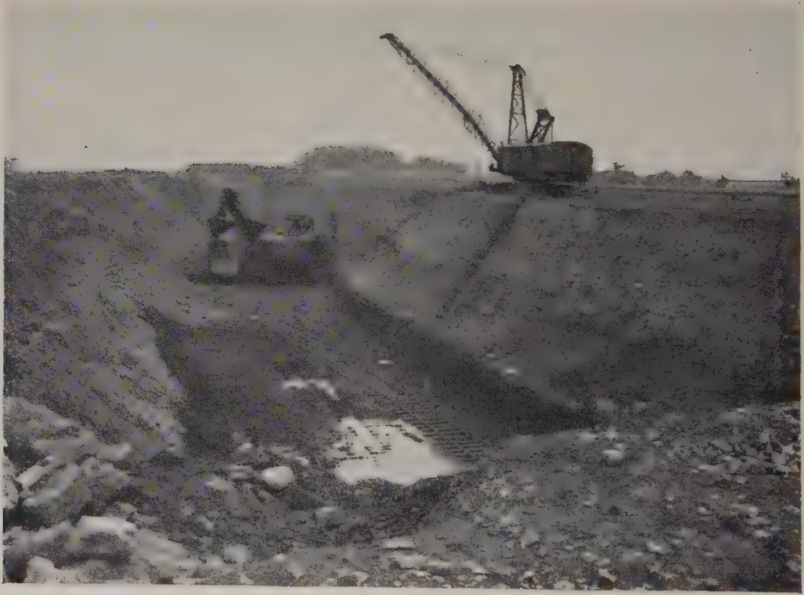


Fig. 4.—Dragline working cover of medium depth. Baring consists of limestone overlying clays. The limestone is dumped to form base of heap. Face shovel is loading the bared iron ore into rail wagons.

and respreading on, worked land levelled to receive it. Ideally, to avoid double handling, after the initial development, there should always be sufficient land levelled to receive the next stripping of topsoil.

Where limestone is present in quantity in the overburden an effort is made to dump it towards the base of the spoil and to use the underlying clays as a cover. When levelled these provide a better base for the soil and for future farming activities and also prevent the inevitable loss which occurs with soil spread on relevelled rubbly limestone in the cavities formed (Fig. 4).

One of the advantages of indirect restoration is that it can be carried out to a planned profile by delaying operations until a sufficiently large area is available and by making sectional levellings to determine the profile. It is also possible to forecast accurately the future trend for further restoration and to determine the finished contour of the whole area. The effects of bulldozer equipment running over the clays to carry out the programme, however, in many cases tends to cause 'panning', that is, the compacting of the top few inches of the spoil into a smooth hard finish. Replacement of topsoil on this finish may result in soil erosion in dry windy periods since there is little adhesion between it and the clay. The liability of ponding in wet weather is also present, the water having little chance to percolate into the lower level of restoration. Deep-rooting or scarifying of the top few inches of the levelled land before the spreading of the soil helps to cut down these dangers by helping the soil to mix with the broken clay and at the same time giving surface water a better opportunity to percolate.

Frequently, in these areas only a small thickness of soil covers the limestone overburden and as the cover is predominantly limestone the finished levelling is mainly a fragmented limestone surface. The covering of this directly by topsoil would be entirely unsatisfactory, resulting in a loss of precious soil and in land which would not maintain pasture or have any arable use. To prevent this care is sometimes taken to introduce a layer of 18 in. to 24 in. of the base clays as a cushion before the respreading of the soil, thereby giving a much wider scope for the after-treatment programme. Such work requires prior approval from the Ministry and is subsequently paid for from the Restoration Fund.

Deep Overburden

Until comparatively recent years the machines responsible for baring in these areas were shovel-type excavators which, where the area to be worked exceeded the maximum working depth of the machine, were assisted by auxiliary machinery, such as smaller draglines located on the spoil side to deal with any excess material, or by scraping machinery to remove excess overburden. In formulating consent conditions for deep overburden areas the Minister made allowance for the fact that it was usually uneconomic to strip off topsoil from quarries of considerable length and either immediately or later to carry it round the ends of the quarry for respreading on an equivalent area of levelled land. Consequently, the conditions stipulated that except for a specified distance, usually up to 500 yd from the swinging end of the pit, it was not necessary to strip off the soil.

The shovel-type machines stand on the top of the rock bed itself and scoop off the cover with a forward action, in contrast to the dragging action of the other machines. They are limited in the height and radius at which both excavating and dumping take place because of their rigid booms. The dumps form a set pattern following and parallel to the lines of excavation, the dumping height giving a ridge line, forming 'hills and dales'.

Where the shovel alone can deal with the depth of the overburden it is sometimes found that because of the percentage of swell which occurs with the excavated material insufficient space is available on the spoil side to receive it. In this case, to enable the shovel to dispose freely of each 'bite' a small dragline is operated on the heaps to pull back the surplus (Fig. 5). In carrying out this operation opportunity is taken to

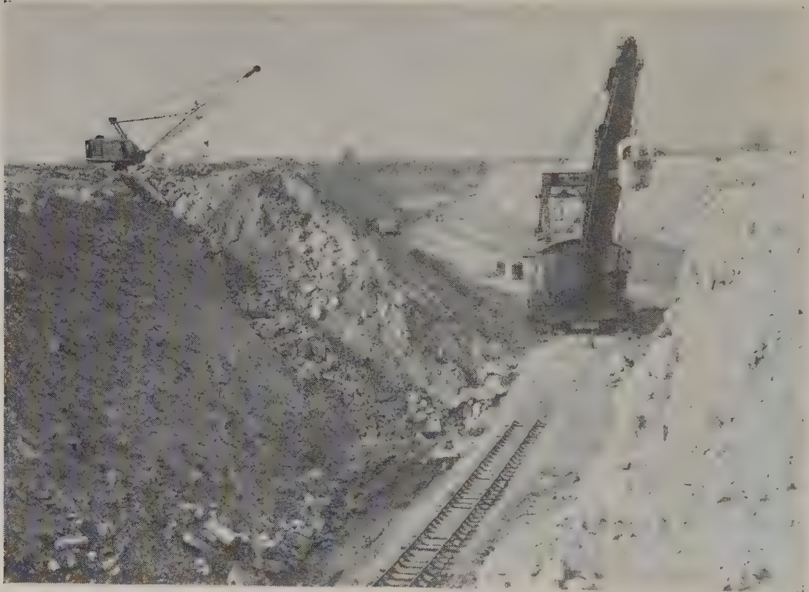


Fig. 5.—Shovel excavator baring in heavy limestone overburden. The small dragline on the spoil side is engaged in pulling back dumped material to assist the main baring machine.

spread the spoil over the hill and dale, leaving a fairly level surface. No charge towards the cost of the final restoration is attributable to the main baring operation, but a percentage of the small machine's time is taken up with levelling off and is therefore allowable as a charge.

During recent years very large walking draglines have been introduced into the orefield, capable of dealing with 75–100 ft of overburden without assistance. Owing to their weight—1500 to 1750 tons—it is essential for the machines to stand on a firm base during operation, and invariably they are found working in conditions where fairly heavy limestone is present. In these conditions the overburden is excavated in the following way: (1) where required the topsoil is stripped separately by a scraper unit and either stored in a convenient place or carried round the end of the quarry and immediately respread on an equivalent area

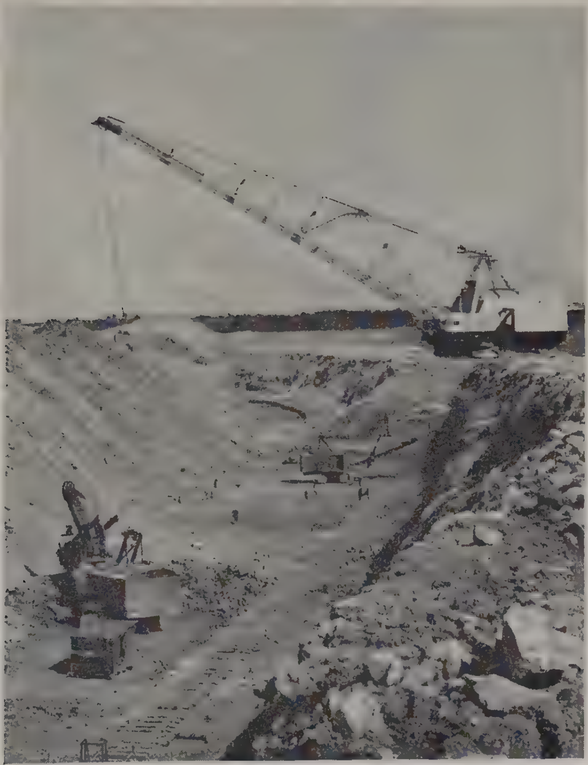


Fig. 6.—Large dragline standing on limestone bed stripping off limestone and lower clay beds from ironstone. In background upper clay beds have been stripped by the same machine to its own level. All the spoil is transferred across the quarry to form dumps ready for levelling by ancillary equipment.

of levelled spoil; (2) the dragline, standing on the limestone strata, takes off sufficient overburden down to its own level within the range of its excavating radius and transfers the spoil directly across the quarry to the dump side, filling over previously dumped spoil formed by the lower baring; and (3) the machine excavates the remaining overburden down to the ironstone, forming a new series of dumps of spoil in the site of the earlier excavation of the ironstone bed (Fig. 6). This is a repetitive process and it will be seen that the limestone is always dumped as a base for the material immediately superimposed on it. On completing the operation of covering the main spoil dumping by the top lift, bulldozers complete the levelling and grading, either in readiness for topsoil or for the commencement of agricultural treatment.

With deep overburden, problems similar to those connected with the restoration of the less deep cover areas occur, with the added difficulty of after-treatment of non-topsoiled land. Because of settlement of the restoration, five years is considered the minimum time before any drainage scheme may be attempted.

Payment for the restoration work is made by the Ministry of Housing and Local Government out of the Ironstone Restoration Fund, according to Section 9 of the Mineral Workings Act, 1951. It is necessary, under Section 10, for the operator to submit an application to the Minister for the establishment of a claim rate. The claim is based on an actual cost incurred in restoring an area within the consent and also refers to the area still remaining to be worked. On determination by the Minister the rate established applies equally to the old and future areas of the consent upon which application will be made. Other than escalation for variation in operating costs, no alteration can be permitted unless it is agreed by the Minister. Periodic claims for actual payment at the determined rate are made as further areas are restored in that consent area, under Section 12; at least six months must elapse between claims.

Claims are met on the basis of their conforming with the consent conditions, the local planning authority being the responsible body for enforcing compliance and the Minister requiring notification of such compliance. There are, however, occasions when works beyond those required by the conditions would improve the restoration effectively. Subject to the receipt of prior approval from the Ministry to carry out such work, a grant of full payment of the cost is made under Section 18(2) of the Act. An example of this is the rooting, referred to earlier, to break up 'panning'. Subsoiling has been accepted as a claim and a grant made under Section 20 of the Act. Every such case is viewed in the light of the benefits to agriculture, taking into account the finance involved.

Prior to the operation of the General Development Order, 1950, No. 1177, certain areas were left in a rough condition, the operating company having made payment to the owner under the lease in lieu of restoration. Provision was made in Section 16 of the Act for local authorities to cause such areas to be restored, the authorities being reimbursed from the Ironstone Restoration Fund.

AFTER-TREATMENT AND USE OF RESTORED LAND

Restored land can be divided into two types—topsoiled and non-topsoiled. The existence of even a thin layer of two or three inches of soil makes an appreciable difference to the germination and development of a crop. The problems of treatment and after-use of such land are therefore considerably less than in the case of non-topsoiled land.

In areas with very shallow cover, where the overburden consists mainly of soil and subsoil only, very little time elapses before the worked land is again satisfactory as a seed bed and it is not unusual for such land to be again growing crops within weeks of being worked for ironstone. With areas of cover of medium depth, however, this is not the case. The material upon which the topsoil is replaced varies enormously in the different localities. Topsoil that has been stockpiled for a period to await respreading may become physically altered, but the long-term effects are not detrimental. Also, heavy clay may be the base upon which the soil has been respread. In these cases it has often been found advisable to sow to a ley, which gives the land time to settle and helps to break the panning effect which may be present, as described earlier, by root penetration. In these conditions drainage is difficult and some scheme is usually required after settlement takes place.

It is not difficult to select a suitable crop, the presence of the topsoil permitting alternate husbandry to be practised. It is, however, important not to over-cultivate, particularly at first, as too deep cultivations may result in the bringing up of some of the rawer clays and large stones hidden beneath the surface of the levelled clays. A fairly heavy application of fertilizer is generally advised, especially in the early stages and with a ley first crop. The promotion of a vigorous growth of grass roots helps break the subsoil and tends to turn any pan into a more friable material.

Non-topsoiled land is much more limited in its potential cropping. In the East Midlands the restored land is most likely to be of chalky boulder clay or estuarine material, the latter sometimes associated with limestone with either the clay or the limestone predominating. The presence of stone is a limiting factor on the type of crop that can be grown. For all practical purposes on this type of land the spoil contains no humus or soil organisms and lacks structure, and natural drainage is poor. Tile drainage in many cases has been installed later—with mixed results. Adequate porous material on top of the tiles is essential, but even so it is often difficult for the water on the areas between the drains to percolate to the tiles.

Experiments were made to discover the most suitable crops for such land and lucerne and lucerne mixtures have been found to crop well. The lucerne, however, died out quickly in three to five years. Cultivation was a problem, particularly with stone present, and non-soiled areas are invariably put down to grass. A 30-lb/acre mixture of timothy, meadow fescue and white clover has been found suitable. The regular application of fertilizer is essential to the success of the grass, the amounts varying with the type of spoil—the help of the Ministry's advisory service is invaluable in these circumstances. A heavy application is usually applied—6 to 10 cwt/acre, aided by basic slag where necessary, in the first instance, followed by 3 to 6 cwt/acre each subsequent Spring. Overgrazing of the land must be avoided and it is important that stock be kept off the land during wet weather to avoid additional drainage problems.

One of the difficulties in putting down stony land to grass is that because of the stone the land cannot be ploughed to give an improvement to the grass. In 1962 facts were disclosed of a new grass weedkiller which looked like making a very advanced contribution in the fight to improve pasture without ploughing. The manufacturing company claimed that a single application was effective in controlling a wide range of grasses and other weeds. In trials its action proved to be somewhat selective; cocksfoot showed initial scorch, but recovered well; ryegrass suffered more with a slow recovery, but white clover recovered well; old meadow grass was killed off. Spraying and oversowing can be done the same day. The price of the weedkiller was quoted at £6 10s. per gallon, stated to be the amount required per acre in average circumstances.

It will be seen that it is sometimes necessary for there to be a change of farming operations on restored land—from arable to cultivation of grazing land.

With the return to agricultural operations the replacement of field boundaries also becomes necessary and the opportunity of obtaining fields of better shape and size is taken. The boundaries are usually replaced with young quicks protected from cattle and making a stock-proof fence by the erection on both sides of a post and bulwark wire fence. A curtain of rabbit netting is often added where the presence of rabbits and hares endangers the quicks (Fig. 7).

Section 20 of the 1951 Act especially caters for the problems of the rehabilitation of the land and grants from the Restoration Fund are made to cover the expenses incurred in the problems referred to earlier. The operation of these grants is in the control of the Ministry of Agriculture, Fisheries and Food. The powers to make payments to owners

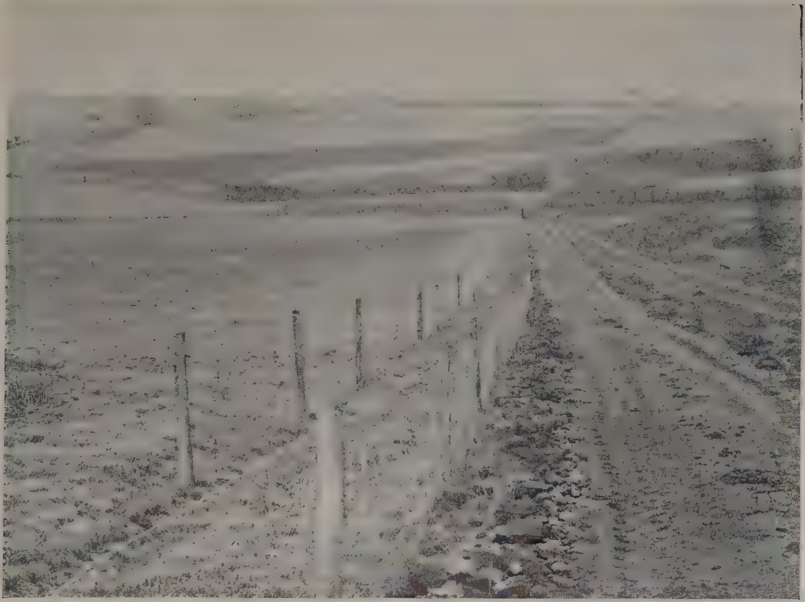


Fig. 7.—A reinstated fence line consisting of new planted quicks guarded on both sides by a post and bulwark fence. The bulwark is hung with rabbit netting as a further precaution. To the right of the face is a reinstated field access road.

and occupiers in respect of agricultural operations or new fixed equipment needed on the restored land are vested in the Minister. The Land Service of the Ministry under the Regional Land Commissioner, working in close co-operation with the National Agricultural Advisory Service, carries out the work involved in investigating and advising on claims.

Prior to the Act of 1947 many of the worked areas in conditions of deep overburden had been left as 'hill and dale' and in post-Act times it has also occasionally been found desirable from the point of view of planning to leave such areas. The most satisfactory method of dealing with these areas is to afforest them. It will be appreciated that the 'hills' varied both in their constitution, that is the amount and size of stone, and in the mixture of soil and subsoil present. From a practical point of view the conditions can be classified as (1) heavy boulder clay which, after heavy rains, may hold water badly in the 'dales' or hollows; and (2) lighter soil areas, consisting of sands, estuarine clays and friable limestone.

From experience it has been found that common alder (*Alnus glutinosa*) is the only crop to grow well in the former areas. Oak (*Quercus pedunculata*) has been grown, but its progress is slow. In the second category

European larch (*Larix europaea*), sycamore (*Acer pseudoplatanus*), Scots pine (*Pinus silvestris*) and Corsican pine (*Pinus laricio, var. corsicana*) have been most successful.

The establishment of an area is carried out by first bulldozing 'rides' across the hill and dale banks a chain wide and 200 yd apart. These facilitate the later extraction of thinnings and final crop and form firebreaks. The perimeter of the area is wired against rabbits and planting is carried out at five-ft by five-ft spacings, about 1400 plants to the acre being required. The bottom of the dales are left free to facilitate the extraction of the thinnings, utilized for fencing material on the estates and quarries, and the second-grade materials in the steel works for loading strips.

It will be seen that such afforestation puts otherwise derelict land to good use, besides improving the 'amenity value' of a district. The woodlands themselves are dedicated under the Forestry Commission schemes and qualify for the payment of grants. In addition, qualification for a Planting Grant—at present a flat rate of £35 per acre—is possible under the 1951 Act.

Although afforestation is not site restoration, it has been referred to since it confirms that every effort is made to utilize what might otherwise have been derelict land.

Discussion

(Papers 24–27)

Professor J. H. Taylor,[*] introducing Papers 24 to 27, said that they were concerned with the general subjects of stripping and restoration. The four papers all dealt to some extent with the performance of shovels and draglines. Mr. Dover, in his paper 'Stripping of rock overburden', drew on his long experience of problems of stripping in the bedded ironstone field of South Lincolnshire, where a considerable amount of the overburden was solid limestone. Cut-and-fill stripping was the common practice and Mr. Dover discussed the criteria that determined the type of excavator, its dump radius and its bucket capacity. Both shovels and draglines had been employed and the variations necessitated in blasting technique and in benching for the two types were described.

The successful employment of a large walking dragline in heavy limestone overburden was noteworthy since in a recent admirable survey of open-pit mining practice in Canada[†] one read that the use of power shovels was 'almost mandatory where the material to be stripped is rock that must be drilled and blasted'. However, in Mr. Dover's experience, under such conditions working costs did usually favour the face shovel.

Attention was also paid in the paper to the possibility of using truck haulage or belt conveyors for the disposal of overburden and, in a final section on instrumentation, two types of continuous recorders of operating times, in use on the United Steel Companies' shovels and draglines, were described.

In 'Economic comparison and detailed discussion of dragline versus shovel for removal of overburden' Mr. Zeindler was also concerned with the relative economics of shovels and draglines in stripping operations. Perhaps the speaker could be forgiven for emphasizing the point that, more than any other single factor, geological conditions affected the choice of equipment—such factors as character of overburden or the relative regularity and continuity of the ore bed. Mr. Zeindler presented two sets of figures of output and operating costs, shovel versus dragline. For one example he took an imaginary small operation worked by 7-yd^3 machines; for the other a much larger project employing machines of comparable capital cost, one a shovel with dipper capacity of 45-yd^3, the other a 35-yd^3 bucket. In both operations the shovel had the edge on costs, to the tune of about one-third of a cent per cubic yard. In a final section the author summed up the advantages and disadvantages of the two types of machine. On the one hand there was the greater reach and dumping radius of the dragline, its greater manoeuvrability, capacity to 'chop down' if necessary (although considerable loss in efficiency was involved) and its ability to strip the uneven top of an irregular orebody. On the other hand, the greater capacity of the shovel, its positive action, shorter cycle time and lower drilling and blasting costs had to be taken into account.

---

[*] King's College, University of London.

[†] Dubnie, A. Open pit mining practice in Canada. *Miner. Surv. Can. Dep. Mines* 4, Ottawa, 1964, p. 19.

In 'Use of walking draglines for intensive opencast coal and ore mining' Professor N. V. Mel′nikov and Engineer N. N. Mel′nikov outlined the range of walking draglines available in the U.S.S.R. It would undoubtedly be of great interest to compare the specifications of their machines with those used in the West. The operations from which the authors drew their examples were mainly coal, iron ore and manganese workings.

In addition to the use of draglines for stripping, their employment for driving haulage and drainage cuts, for constructing spoil banks, for stockpiling and for loading was discussed. The statement (p. 583) that within ten years three-quarters (by bulk) of the Soviet Union's mineral production would come from opencast operations was perhaps an encouraging indication of the timeliness of the present Symposium.

The final paper, 'Restoration of ironstone workings in the Midlands of Great Britain', dealt with the general problems of restoration in bedded ironstone fields. The figure of 15 500 acres as the total area worked or in process of being worked for ironstone showed the magnitude of the problem. The statement that 85 per cent of the worked-out ground was now usefully employed indicated the success with which it had been met.

After dealing with the historical and legal background of land restoration, Mr. Jones went on to discuss the factors involved in areas of shallow and of deep overburden. While it was customary to strip and respread topsoil in shallow operations, that was often uneconomic in the deeper opencasts. Massive blocks of rock, such as those from the Lincolnshire Limestone, presented a perennial problem and had, if possible, to be kept well below the final restored surface.

It was perhaps worthy of comment that none of the four papers paid much attention to the tractor, described elsewhere as 'perhaps the most versatile piece of equipment for earth-moving and stripping'.*

At first sight it might seem that the fourth paper on restoration differed in scope from the others under discussion, but he thought the difference apparent rather than real. It was generally agreed nowadays that restoration should be treated as an integral part of opencast operation. Machines used for restoration were commonly used also for stripping and indeed, as both Mr. Dover and Mr. Zeindler indicated, restoration requirements were often among the factors influencing the choice of stripping methods.

Monsieur J. J. Danjoy† said that Mr. Dover gave very interesting data regarding the choice of stripping equipment, but that choice itself depended essentially on the overburden ratio in connection with the topographic limits of the opencast mine. When planning the introduction of stripping equipment on a deposit, it was necessary to investigate the economically advisable stripping ratio and to fix the limits of the quarry. It supposed that the part of the deposit outside those limits would be either mined by an underground method or abandoned.

Mr. Dover's views regarding the methods of determining those limits and, if possible, information on how his own Company had dealt with the problem, would be of great interest. Could he also state the admissible stripping ratio in the particular case described?

Professor E. Teply‡ said that in Paper 25 (p. 571) the maintenance costs were given as $25.65 for the 7800 dragline and $38.57 for the 5561 shovel. In his opinion, if the very large expenditure incurred in relation to ropes (they were about three times as long for draglines as for shovels and wear was much greater) were taken into account, the total costs would not favour draglines, and costs per cubic yard might be found to be greater for draglines than for shovels. Moreover, the direct operating costs data were based on the scheme detailed in Figs. 3 and 4 (p. 570), where maximum operating heights of 56 ft for the shovel and 83 ft for the dragline were accepted. He thought it would be very interesting to calculate the costs per cubic yard on the basis of equal operating heights, for instance for 83 ft. In that case a supplementary dragline on the top for the excavation of the remaining height of 27 ft would be necessary. If the amortization costs were

---

**Op. cit.*, p. 15.

†Société Française d'Etudes Minerais, France.

‡University of Zagreb, Yugoslavia.

added to the direct operating costs it became evident that the use of a dragline would be far more rational than the use of a shovel.

It would be very interesting to calculate for all possible spoil heights so as to determine the break-even points for the economic application of either shovel or dragline.

In comparison with conditions in the U.S.A., in Yugoslavia the cut-and-fill stripping method for coal seams of 20 m and over had been introduced about ten years ago and the M 7200 and 7400 Marion draglines have been in use since then. Because of the special conditions in the mine of Banovići in Bosnia—small opencast workings in various places between faults—those could be operated by draglines only, as could be seen from Fig. A.

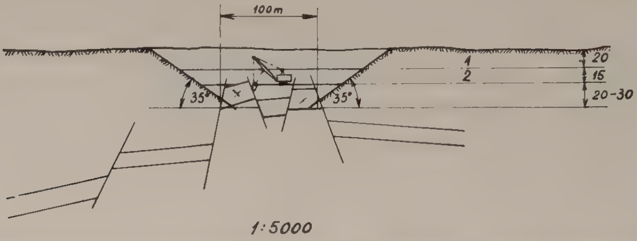


Fig. A.—(1) Overburden removed by shovel; (2) Cut-and-fill stripping by dragline.

In other sections of the same mine, where the coal seams were inclined from 15 to 25°, the same method as that described in Paper 26 (p. 582) was used but, in view of the large expense involved in dragline operations, especially with regard to the high amortization costs, modified as described below.

Large draglines were used for spoil stripping only. Coal production was carried out in cross-laid longwalls 20 to 30 metres in height. By successive blasting the coal was cast down and loaded by tractor shovels on to chain conveyors placed along the longwalls. The coal was then hauled by the chain conveyors up to the main belt conveyor in the roadway. By that horizontal roadway and by an inclined

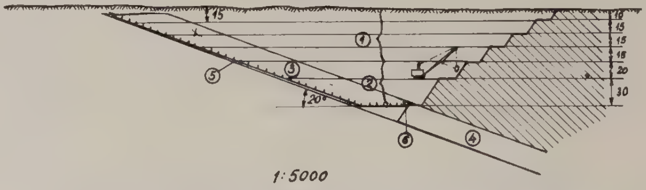


Fig. B.—(1) Overburden removed by shovel; (2) Cut-and-fill stripping by dragline; (3) Coal seam for opencast excavating; (4) Coal seam for underground exploitation; (5) Chain conveyor; (6) Belt conveyor in the main road.

drift at the end the coal was hauled to the surface of the open pit. Thus in working and hoisting coal the heavy dragline had been replaced by much lighter equipment. Fig. B illustrated the method, as a cross-cut vertical section.

Professor J. H. Taylor asked Mr. Dover to what extent the decision to operate the Exton Park area with its heavy limestone overburden by means of a W 1400 dragline had been influenced by the demands of restoration.

With regard to the paper by Mr. Jones, he noted that restored areas which were not topsoiled were now invariably put down to grass. He enquired how long it was before such land could be stocked as it was prior to working.

Mr. A. Shadmon* said that his experience with the use of rippers in the quarry industry revealed that the capacity of really large-size machines was too large for a medium-size quarry. Since the operation described by Mr. Dover was concerned with a much larger than average quarry he wondered if any information on a ripping device was available for the removal of rock overburden.

Mr. R. Weisskircher† said that in comparing the relative performance of draglines and shovels in loose material the parameters concerned were the height of the bank and the distance the material had to be transported. When dealing with hard-packed or interlocked rock, however, further factors were involved—digging capacity, the possible need for supplementary equipment to improve the 'fill-factor' of the bucket, blasting or ripping. Those naturally led to increased costs. The resistances involved were overcome by the shovel and dragline in quite different ways—and with quite different results.

If Mr. Dover and Mr. Zeindler could give details with regard to the economic effects to be expected of the above factors, so that a choice could be made of either shovel or dragline, based on theoretical considerations and proved by actual operations, he would be grateful.

Mr. E. Rich‡ asked Mr. Dover for further information with regard to the reduced maintenance mentioned in the section 'Design of Excavating Machines' (p. 564). He noticed the reference to the use of better materials, and an indication of what those materials were would be of considerable interest, particularly insofar as they concerned abrasion resistance—always a major problem on earth and rock-handling equipment.

---

*Ministry of Development, Jerusalem.

†Ingenieurbüro für Bergbaufragen, Dr. Ing. Otto Gold, Cologne, Germany.

‡International Nickel, Ltd., London.

Mr. R. F. Lethbridge* said that in England land restoration had become a vital part of opencast mining under the control of the Town and Country Planning Acts. Everyone in the United Kingdom was very much concerned with the problem and, with the larger volume of material actually taken away, that was of a different nature to alluvial mining for tin and gold where the amount actually removed was negligible, giving rise to problems of swell.

In the past, in Malaysia one of the great arguments against granting more land for mining purposes was the general statement: 'You miners take the land and spoil it'. He thought that had been very nearly true.

In later years, since the war, there had usually been a condition in leases demanding some measure of levelling of worked-over ground. The sandy hills left behind, however, were not considered to be suitable for agricultural use, although that could be shown to be not strictly accurate.

There were certain factors which stopped the mining industry taking much interest in restoration—one being the refusal of the Malaysian authorities to allow re-conversion from mining title to agricultural title. Under those conditions there was no advantage to the miner in restoring land and much of the worked-over land in Malaysia was therefore, in its present state, useless. That was held against the mining industry even though, in effect, the total area given over to mining was small. Some one per cent of the land area produced 40 per cent of the national income.

Mr. S. W. L. Pardoe,† referring to Paper 27, said that on opencast coal sites in the United Kingdom it had been found that, working at overburden ratios between 10 and 20 to 1, the bulking of the overburden more than made up for the volume of the coal removed. A normal bulking figure after regrading was 12 per cent. He would be interested to know what bulking Mr. Jones had found on his restored ironstone sites and how much settlement of the restored surface he found in the first few years.

---

*Rio Tinto-Zinc Corporation, Ltd., London.
†National Coal Board Opencast Executive.

Mr. J. G. Lancaster* enquired if the continuous recorder described by Mr. Dover for use with draglines was designed by his own Company or if it were available generally. His own experience of Service recorders on lorries had shown them to be almost impossible to interpret at times. Did the instrument described give rise to any ambiguity in interpretation.

The availability of 'action' information on draglines and other earthmoving machinery was of paramount interest to the speaker's own Company at the present time.

Mr. D. G. Hemmant† referred to an opencast operation with which he had been connected in which the volume of overburden occupied less area in the spoil heap after compaction by the continual travelling of dump trucks, bulldozers, etc., than it had occupied in its original position in the cut. First reactions had been that the surveyors must have been in error, but the figures had been checked and appeared to be correct. He wondered whether others present had had similar experience and, if so, whether anyone could offer an explanation or indicate the circumstances in which that super-consolidation was likely to occur.

Mr. D. R. Ward Jones‡ said that he wished to stress the operation of the Mineral Workings Act, 1961, set up by the Ministry of Housing and Local Government under the Town and Country Planning Act, 1947, for the ironstone industry. The Act provided for a restoration fund which made grants (*a*) to the producer who had to bear the first £110 per acre himself before recovering any excess costs and (*b*) to producers or farmers who spent money on hedging, drainage, or other items agreed by the Ministry of Agriculture which they considered would improve the worked land.

All producers contributed to the fund at the rate of 1⅛d. per ton of ironstone, the owner contributing a similar sum, except in respect of land owned by charities or the subject of a pre-1950 lease requiring full restoration. The Exchequer made a further contribution of ¾d. per ton and the fund now stood at nearly £500 000.

He would also point out that the ironstone industry was the only one to be bound by statute for restoration and was proud of its results. He considered the industry to be the only competitor to the National Coal Board in that particular field.

---

*The Fullers' Earth Union, Ltd., Surrey.

†Truscon, Ltd., London.

‡Ore Mining Branch of The United Steel Cos., Ltd., Rotherham, Yorkshire.

Mr. R. A. Gorges[*] said that Mr. Dover quoted (p. 562) a figure for the 'swell', or area of the spoil bank compared with the cut section, of 35 per cent (in the absence of local knowledge as to what it actually was). He understood that to mean that the broken spoil occupied 35 per cent more room than the solid before cutting. Figs. 1 and 2 of Mr. Zeindler's paper (p. 567) showed the volume after dumping to be 25 per cent greater than before.

The difference between those two figures was not what interested him as much as the difference between either of them and the factor which one was accustomed to apply in underground work where, in the Witwatersrand quartzites, it was accepted that 12 ft^3 of solid became 20 ft^3 when broken, a swell of $66\frac{2}{3}$ per cent. A similar factor was applied in other mines. He would be grateful if the authors would comment on that.

Mr. C. O. Frush[†] said he would like to comment on some aspects of the problem of land restoration with reference to the United States. The need for restoration depended to a large degree on the value of the land involved. In the western states, where low rainfall and a relatively low population density, together with difficult topography, made much of the land surface marginal in value, reclamation was largely unnecessary, and would often be wasted. In the Eastern States the land surfaces were more valuable and reclamation was much more widely practised.

Work done some years ago by the United States Forest Service showed that 'levelling' was not necessarily the best technique. Often the land had more value if it were left in a rough condition, particularly if good planning were involved. Not only did rough ground grow a better and more varied timber crop, but it also might be better adapted to beef production. Since much of the Midwest was rather flat, those rough areas, when reafforested, make excellent recreational areas. Some had been converted to State parks.

It should be noted that often much of the pressure for restoration came from people who were uninformed about the mining industry, or even uninterested in it and its problems. In at least one case their interest in land restoration was fostered by a large miners' union; the union concerned had a large membership among the underground coal miners, but had been unable to organize the lower-cost surface mining operations. It apparently viewed the additional cost of surface restoration as a means of reducing the cost advantage of surface operations.

---

[*] Camborne School of Mines.

[†] Colorado School of Mines, Colorado, U.S.A.

Mr. V. H. R. Oliver,* in reply to Mr. Hemmant's request for information on the amount of bulking and compaction of broken materials in waste dumps experienced on other opencast operations, said that at Chambishi Mines, Ltd., Zambia, they had now removed approximately 9 500 000 bank cubic yards from an open pit, of which approximately 8 600 000 bank cubic yards had been placed on a waste dump. That dump has a volume nearly equal to that of the pit and their combined bulking and compaction factor was therefore of the order of 110 per cent of the *in situ* volume.

Materials placed on the waste dump consisted in the main of weathered argillites and quartzites, which were dumped as fairly loose materials by 27-ton capacity trucks, and to a lesser degree of blasted unweathered hard rock lumps of the same geological strata.

During the design of the waste dumps, swell factors (*in situ* volume/broken volume) of from 0·67 to 0·75 were assumed for the materials excavated in the open pit, and it was assumed that they would be compacted by 15 per cent when placed on the dump. They would thus have obtained a combined bulking and compaction factor of 113 to 127 per cent of the *in situ* volume. Their original assumptions were therefore not entirely accurate, but might be more closely approached in time as the ratio of hard rock lumps to loose materials placed on the dump increased as the pit was deepened.

Mr. A. Major-Stevenson† asked Mr. Dover if he could give an output cost per cubic yard for the W 1400 dragline to compare with the figure of 10d./yd^3 given for the Marion 5323 stripping shovel.

---

*Chambishi Mines, Ltd., Zambia.
†Ruston-Bucyrus, Ltd., Lincolnshire.

Authors' Replies

(Papers 24–27)

Mr. T. M. Dover: In reply to Monsieur J. J. Danjoy (p. 603), I agree the main factor in choosing the size of a stripping machine is the overburden–ore ratio although this is largely an economic problem so that value of product enters into our calculations.

If a = value of one yd^3 of ore, b = depth of ore in yards, y = cost of stripping overburden per yd^3, and x = greatest depth which can be stripped in yards,

then
$$x = ab/y$$

If the overburden depth continues to increase above the figure for x, then the possibility of underground mining must be considered. The problem is greatly complicated by uncertainties of percentage extraction underground, which may well be under 50 per cent in ironstone, and lowering of ore quality due to limitations of the parts of the bed which can be extracted.

I believe the decision as to maximum depth workable opencast may have to depend on comparison of overall cost at various depths and overall estimated or actual cost of underground working in the same area. The loss of ore due to low extraction by underground systems and consequent greater area required to produce a given output over a period may well make opencast working more attractive from the national point of view, especially as it can be shown that complete restoration of opencast working can be preferable to subsidence produced by underground working at shallow depths.

At present a ratio of 12:1 overburden to ore can be considered the limit in the Northampton Sands, but this varies considerably from site to site and has increased over the years with better stripping methods.

Professor E. Teply (p. 603) has suggested that dragline rope costs will be higher than those for face shovels. The operating costs (d./yd^3) over several years for a 20-yd^3 dragline and a 17-yd^3 face shovel in similar conditions (given below) confirm the suggestion.

Dragline hoist ropes	0·135	Dragline total rope cost	0·258
Dragline drag ropes	0·123	Face shovel hoist rope cost	0·233

Difference in favour of face shovel 0·025

Professor J. H. Taylor made a valid point in his introduction in stating that restoration must be considered in planning the stripping operation. At Exton Park the decision to use a dragline was largely due to this.

At that time truck haulage or mobile crushers and face belt conveyors for ore were not considered so that it was necessary to plan for rail haulage which for maximum efficiency requires the longest possible working face and a connection for incoming and outgoing trains at each end. Because of this it was difficult to transport the soil and subsoil required for restoration round the pit ends and the dragline is used to transport the soil across the pit from heaps where it has been collected by scrapers on the cut side to heaps on the spoil side from which it is subsequently spread again by scrapers.

Mr. A. Shadmon (p. 605) asked if the large ironstone quarries can use effectively the capacity of large crawler tractors pulling rippers. They have not much application on the thick beds of limestone which have to be stripped since the broken material must be removed before lower layers can be reached by the ripper tyne and this is not convenient with either stripping shovel or dragline operation. Ripping has occasionally proved effective when, after stripping, low-grade ore from the top of the ore bed must be removed before loading. A band of chert in this low-grade ironstone made the drilling of shot holes and blasting an expensive alternative to ripping.

Mr. R. Weisskircher (p. 605) asked about the relative efficiency of face shovels and draglines digging hard-packed and interlocking rock. The face shovel can dig better under unfavourable conditions, but for efficient working neither type should be called upon to do more than dig a loose and displaced pile. This is not always possible as the stability of the face, particularly with a dragline, must be considered in planning the amount of blasting to give the best results.

Mr. J. G. Lancaster enquired about excavator records (p. 607). The Service recorders used on stripping and loading face shovels were not the vibration operated type. They are operated by the closing of a battery-powered circuit each time the bucket catch mechanism is operated. An electromagnet moves the stylus on the chart which is easily interpreted either to read the number of cycles or the operating time. The swing recorders on the dragline were designed to the operating company's requirements and are much more costly.

Mr. R. A. Gorges mentioned (p. 608) the percentage swell varying from 25 to 66 per cent. Very large variations have been noticed in the same overburden face leading to changes in earnings of machine crews paid on output measured 'in place'. This has been traced back to changes in the proportion of 'fines' in the rock. The figure of 35 per cent was found from a number of measurements of spoil banks compared with 'in place' cuts. In limestone underlain by clays local knowledge and judgement would appear to be necessary in assessing swell.

Mr. A. Major-Stevenson (p. 609) asked for comparative costs of particular machines. These might be misleading as utilization due to ore requirements affected the result greatly, but they did favour the face shovel over the dragline.

Mr. E. Rich (p. 605) asked for further information regarding reduced maintenance due to better materials in excavator design. I have asked Mr. A. Major-Stevenson of Ruston Bucyrus, Ltd., to provide this information and the following is his reply:

Manganese steel castings are now widely used where work-hardening properties are called for and this means they are extensively used in the construction of dippers and dipper teeth and on some of the larger types of dragline bucket. These steels have excellent work-hardening properties in addition to a fair degree of resistance to abrasion.

Manganese castings with lower proportions of manganese are frequently used in the manufacture of track links.

High yield steels to conform with BS 968, of welding quality and relatively low carbon content, are becoming increasingly popular for the fabrication of front-end equipment on excavators because of the greater strength for lower weight derived from their use. Dragline buckets will also incorporate steels of this type to a greater extent as time goes by. Other types of steel which are occasionally used for varying applications includes copper–nickel–chrome alloy steel which has a yield strength of 50 per cent greater than normal mild steels, together with the benefits of anti-corrosive properties which make them very acceptable for use on machines and equipment exposed to corrosive atmospheres.

High carbon steel castings with a carbon content of about 0·40 per cent are becoming increasingly popular for the manufacture of gears and are generally favoured when the machined gears are to be flame hardened. It is normal practice for nickel alloy steel shafts to be incorporated wherever possible.

Mr. R. W. Zeindler: With regard to Professor E. Teply's comments (pp. 603–5), the comparative maintenance costs between the large stripping dragline and shovel included the total maintenance costs which, therefore, included the costs of the various ropes or cables involved. A stripping shovel has multiple reeving for its hoist cable and, in fact, has a longer cable than the comparative dragline. It is true that a dragline utilizes an additional cable, the drag cable, and this particular cable is subject to wear. However, it must be remembered that in a stripping shovel of the type discussed there is a great deal of maintenance involved in the crawler assembly and lower frame structure. All of this maintenance is not present in a dragline. As mentioned initially, all items of maintenance were taken into consideration in the comparative costs and thus no material cost changes are needed.

It must be remembered that two standard-type units were compared under general conditions; strictly from an illustration standpoint the operating heights of 56 ft and 83 ft are obtained. If the conditions are varied so that certain restrictions are imposed upon either of the units, then obviously the other remaining unit would be favoured. This was not the intent of the paper, but it was meant to provide an overall comparison and illustration of what can be accomplished with these two general types of stripping units. Many factors can be introduced into any theoretical application—which will naturally affect the final machine selection.

Nor would spoil heights alone determine the so-called break-even point for the economic application of either shovel or dragline. There are extremely long-range front-end combinations available for stripping shovels that materially increase the height of overburden and resulting height of spoil that can be handled. As briefly mentioned above, and detailed in the paper, all of the overall conditions of an application must be carefully weighed before any final and conclusive assumption can be reached.

Professor Teply provides some interesting information on Yugoslavian operations. As pointed out in my paper, the dragline is a much more flexible unit, as it operates on top of the overburden and, therefore, can make adjustments in the geological features that may have contributed to uneven conditions in the mineral deposit. Obviously, a stripping shovel could not be utilized in a highly faulted area where no relatively horizontal plane could be established for the shovel to travel on.

In reply to Mr. R. Weisskircher (p. 605), one of the conclusions mentioned in my paper was that as the material to be removed becomes harder a shovel would definitely be more and more favoured. It has been determined that for the same general conditions of overburden a dragline stripping operation will require 10 to 30 per cent higher drilling and blasting costs than a stripping shovel.

In reply to Mr. R. A. Gorges (p. 608), the swell factor utilized in application studies refers to the overall swell from the in place position in the bank to the dump position in the spoil. There is no question but that when the rock or overburden is being loaded by the dipper or bucket a higher swell factor is present than when the material is located in the spoil bank. Actually, the material undergoes several instances of swell and compaction during a typical mining cycle. The material at first is in place, then drilled and blasted—where a certain swell results. The material is then loaded into a bucket or dipper where, again, a certain portion of it becomes compacted and another portion swells again. The material is finally dumped into a spoil pile where compaction results and final compaction occurs as additional material is placed on top. In time, due to climatic conditions, a certain amount of final settlement and compaction will result.

Mr. T. W. Jones: In reply to Professor J. H. Taylor (p. 605), it is not possible to be definite as to the length of time before non-topsoiled areas which have been grassed down can be stocked to the same extent as prior to working. It could be from 5 to 10 years and even longer, according to circumstances. Good management is vital to bring this time to a minimum, both in the establishment of a good sward and to prevent the aggravation of the poor drainage conditions. The careful consideration to such matters as the application of fertilizers and the withdrawal of stock from the land during wet weather until a successful drainage scheme has been developed will help to reform structure and re-establish humus as quickly as possible. Other difficulties which affect the timing are that the grass does not continue growth for as long a period on this type of land, and also during dry weather it can cease to grow, again necessitating the withdrawal of stock.

With regard to the query of Mr. S. W. L. Pardoe in respect of bulking (p. 606), this is of little consequence in the case of shallow overburdens, but with the deeper ones it is a factor which has to be taken into consideration in calculating dumping room available during baring operations. For overburden ratios of over 2 to 1 a figure of 30 to 35 per cent is used. This has been found acceptable for many years. A recently worked area with an average overburden of 35 ft and a rock extraction of 12 ft has been restored to the same level as the adjacent virgin surface. The amount of subsequent settlement of restored land has not been measured, so far as I am aware, but it has never been so much as to be obvious. Since the areas worked are open lands and are in the main returned to agriculture after restoration, the amount of subsequent settlement has not been considered sufficiently important to justify its survey. However, since the major settlement occurs in the early years after restoration it has not been considered advisable to install a drainage system until at least five years have elapsed.

Mr. D. G. Hemmant referred (p. 607) to the volume of overburden occupying less area in the spoil due to compaction by the continual travelling over the spoil of the machinery used in dumping than it did in its original position. This has not been the experience in the operations with which I have been connected since the bulk of the spoil has been dumped by a face shovel or dragline type of machine and only the top portion of this has been run over by auxiliary equipment such as Mr. Hemmant quotes. This has tended to lower the final finish by a few inches only by leaving a 'panning' effect which, in any case, is ultimately broken up by either deep-rooting or scarifying by machinery or by the natural process of root development during agricultural operations.

GROUP IV—OPENCAST MINING
(bedded deposits and general questions)

Session 8—Operations (Papers 28–30)

Chairman: Mr. G. C. Payne *Rapporteur:* Mr. D. G. Hemmant

622.271.4:622.333(428.2)
622.271.3:622.333(413.3)

Paper 28

Opencast Coal Mining in Great Britain: Operation of Acorn Bank and Westfield Sites

W. G. MARTIN, M.C., A.M.I.C.E., A.M.I.Struct.E.
Director, Costain Mining, Ltd.

SYNOPSIS

The two large sites of Acorn Bank and Westfield are at present in operation under contract to the National Coal Board Opencast Executive.

Acorn Bank is a strip-mining site where the coal is obtained from a series of parallel cuts. The spoil overlying the coal seams in one cut is cast into the cut previously excavated, from which the coal has been recovered. The average width of cut is 80 ft, and three or more seams are worked, the depth varying from 20 to 220 ft. When in full operation, two 1150B draglines were used for overburden removal. Coal was raised from the cut by four high-speed derricks.

The contractual output was 16 500 tons of coal per week.

Westfield, on the other hand is an open-pit mine, more comparable with a normal quarrying operation. The overburden and partings between the coal seams are excavated by face shovel, crushed, and conveyed by high-speed belt to a tip 1¾ miles from the excavation area.

The contractual output is 13 000 tons of washed coal per week.

Some details of plant output are given.

On both sites AN–FO explosive is used for overburden preparation, but at Westfield, because of wet holes, slurry blasting is now being developed.

The different methods of payment pertaining to the two sites are explained.

ACORN BANK STRIP-MINING SITE

The Acorn Bank pit in Northumberland was opened up in 1955. Three seams of coal extend over the whole of the 440-acre site. The High Main seam is excavated in one leaf and has an average thickness of 39 in. The Top Grey seam is in two leaves, separated by a parting of thickness varying from 1 to 4 in., the two leaves of coal, each about 15 in. thick, being dug separately. The Bottom Grey seam has four leaves of coal with a total thickness of about 84 in. The parting between the top two leaves of coal is several feet thick in one part of the site, while the remaining partings are normally only an inch or two in thickness. Two other seams, the Top Yard and the Yard seams, have been worked in one area of the site between upthrow faults which have prevented their recovery by the deep mines operating in the area.

The distance between the three main seams varies considerably over the site. In the north, all the seams are contained within a depth of 30 ft, but in the southeast the seams are as much as 60 ft apart.

Initial excavation

Excavation was started at the eastern end of the site, where the coal seams were deepest. This is the reverse of usual procedure, but the method was imposed as a condition of the contract, with a view to ensuring that the site could be restored progressively.

The box-cut was 3200 ft long by 230 ft deep and was 100 ft wide at the coal surface. This first excavation of 10 000 000 yd^3 provided the spoiling room necessary for the normal strip mining which was to follow.

Drift excavation

A layer of drift material covered most of the site to an average depth of 30 ft. It consisted of boulder clay overlain by a yellow clay, and in many places there was a bed of sand between these clays. The drift was excavated by three 120 RB 5-yd^3 electric face shovels and loaded into 22- or 27-ton dump trucks. Where the bed of sand was thick the sand and yellow clay were dug separately and stockpiled for later restoration as subsoil. The remaining material was run to tip over the de-coaled area. Drift excavation is now substantially complete, but much of the work described below continues.

Rock excavation

The overburden below the drift consists of shales and sandstone which are too hard to excavate without blasting. Joy 58 BH Champion drills, with tricone bits, are used for drilling $6\frac{5}{8}$-in diameter holes into the overburden, usually on an 18-ft square grid. The holes are charged with an ammonium nitrate–fuel oil mixture primed with 10 per cent gelignite, the charging ratio being $3{\cdot}5\ \text{yd}^3/\text{lb}$. The use of platform charging enables an attempt to be made to place the bulk of the explosive in the heavy sandstone bands.

This material also is excavated by the three 120 RB face shovels, down to the dragline/derrick bench level, approximately 90 ft above the top coal seam.

The main excavation has been carried out by two 1150B draglines, one having a 215-ft jib and a 22-yd^3 bucket, the other a 180-ft jib and a 25-yd^3 bucket. The dragline with the longer jib operated from the centre to the northern end of the coaling cut, where the coal was deeper. The other operated from the centre to the southern end of the coaling cut. Mechanical breakdown could put the machines out of step, and any phase difference of more than a week or so was corrected by adjusting the position along the length of the cut where the first machine to finish the old cut began the new cut. The average width of cut was about 80 ft.

Operating sequence on dragline bench

The sequence of operation of the dragline/derrick bench was as described below.

(1) While the dragline was excavating the end of the previous coaling cut, two derricks followed it along the high wall, lifting out the coal as it was dug by face shovels in the cut below.

(2) A drill followed the derricks along the high wall and the overburden of the next cut was drilled and blasted, firing taking place after all the coal had been removed from that section of the previous cut immediately below.

(3) The dragline moved to a position about mid-way along the cut and excavated down to the top seam of coal, working the half-length of the cut. The derricks followed lifting out the coal.

(4) The drill went down into the de-coaled cut and drilled down to the next seam of coal; the holes were charged and fired.

(5) The dragline moved on to the spoil bank, travelled back to a position opposite the centre of the cut, and dug down to the next leaf of coal. During this operation the machine also dug the rehandle material resulting from (3).

With this system the dragline was employed for removing the thicker parting. Normally, the parting between two of the seams was small enough to be removed by face shovel and dumper, and this was achieved during the night shift using the coaling excavators. The spoil was run to the de-coaled end of the cut.

Accurate programming is essential in operating a site with three main leaves of coal to avoid standing one machine while others complete their section of the work. The major items of plant are, of course, the two draglines, and all the other work was planned to keep these two machines fully employed.

Coal hoisting

A major restriction in the operation of the site was the necessity for progressive restoration, and this precluded the use of a permanent access ramp into the cuts for coal removal. One of the methods originally considered for providing a route to the coal was a twin access tunnel formed in precast concrete units through the spoil banks to the bottom of each cut. The tunnel was to be extended as the cuts moved progressively across the site. This method of operation would have been difficult because the seams of coal were at different levels, and hazardous because deep mine workings beneath the site might cause subsidence.

The method finally adopted called for four high-speed electric derricks (Fig. 1) travelling on rails along the top edge of the high wall.

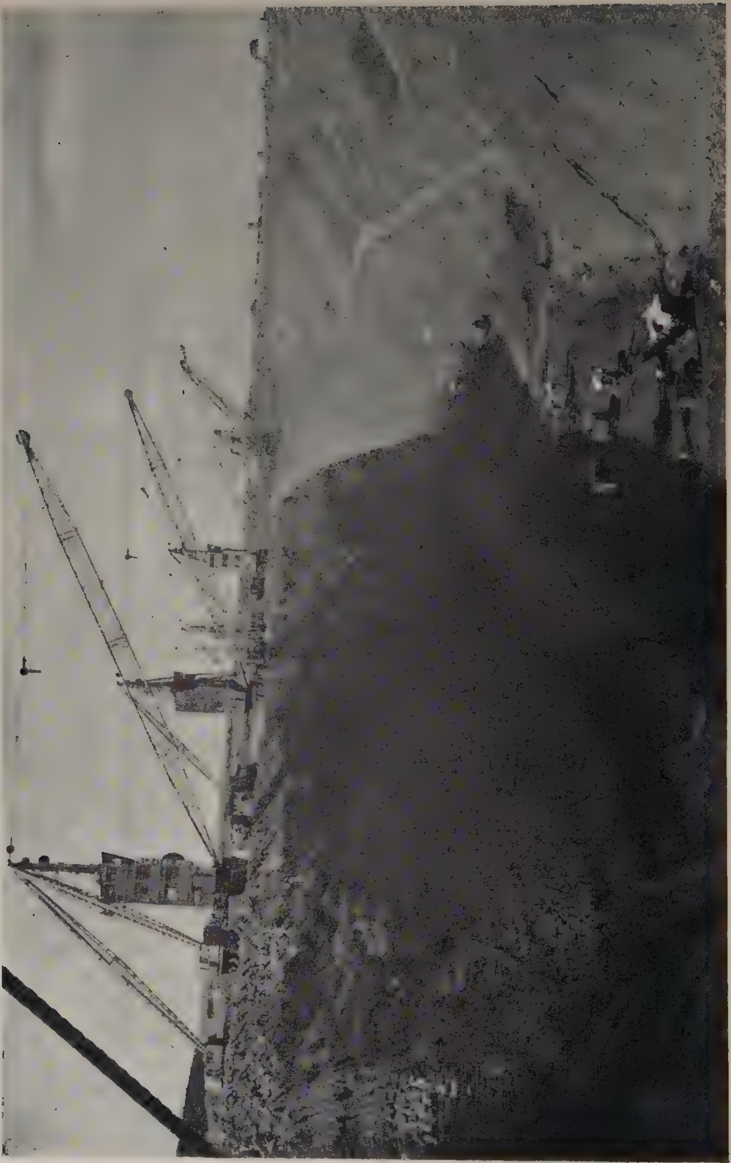


Fig. 1.—Acorn Bank: coal loading from the Bottom Grey seam. In the foreground a $1\frac{1}{2}$-yd^3 shovel is loading a derrick skip. The nearest derrick is raising a full skip. In the background a 1150B dragline is stripping overburden.

The derricks were double-drum type with jibs 152 ft long, and were capable of lifting 13 tons at a maximum radius of 140 ft. The driver's cab was situated near the top of the mast and the derricks had separate hoisting, luffing and slewing motors, and could make all three motions simultaneously. Each derrick operated with two or more coal skips of 10-ton capacity. The derrick lowered an empty skip into the cut and this was swung by hand into a position near a face shovel excavating coal and unhooked. A full skip was then picked up with the main rope attached to a bale bar which lifted just forward of the centre of gravity of the skip, and the second rope was attached to the tail of the skip. The full skip was lifted out of the cut and swung into position over a 40-ton coal hauler. With the tail rope held, the main rope was lowered and the coal emptied directly into the coal hauler without the use of a banksman or hopper. The derrick drivers quickly became skilled in this operation and achieved outputs of up to 150 t/hour.

At one time an attempt was made to operate the derricks using 5-ton capacity grabs to pick up coal from stockpiles formed by excavators on the floor of the cut, but this method was too slow and was quickly abandoned.

The derricks travelled along the edge of the high wall as the cut was de-coaled and were then moved back at right angles to the cut and re-positioned for the succeeding cut. To enable this movement to be made the derricks were fitted with special bogies, each equipped with four 25-ton hydraulic jacks. Each bogie was lifted in turn, when standing on a special crossing rail. The wheel housings were then rotated through 90°, the bogie was lowered back on to the cross-track, and the derrick could then travel at right angles to its original tracks.

The major difficulty met in operating this system of raising coal was the need to work the derricks very close to the edge of the high wall in order to be able to reach into the cut; for such heavy derricks a stable high wall was essential. Owing to the frequent faults occurring on the site, cracks in the high wall were not uncommon, and the cracks were extended by heavy blasting. The drilling pattern and the sequence of delay detonators were adjusted in order that the high walls should be disturbed as little as possible by the blasting for the adjacent cut. It is a tribute to the skill of the foremen and operators concerned that the derricks were operated for many years without an accident.

Current operations

The greater part of the site was worked by the methods described above. The remainder of the site is too restricted to allow two large draglines to be used, and only one dragline and two 120 RB face shovels are currently employed. Similarly, the use of the derricks ceased when the shallow end of the site was reached, but it is planned shortly to re-employ one derrick on a part of the site where access by ramp to the coal seams would be difficult.

Coal excavation

The surface of each coal leaf is cleaned by bulldozer as it is exposed. Final cleaning is carried out by hand to ensure that the minimum of coal is wasted. The coal is excavated by $1\frac{1}{2}$-yd^3 and $\frac{3}{4}$-yd^3 face shovels, each fitted with a bucket having a flat lip, designed for clean loading of the coal. Since the bottom leaf of the Bottom Grey seam is usually too hard to dig without blasting, this coal is normally drilled with a small hand-held compressed-air-driven auger, drilling holes at 4-ft centres; these are charged with 2 oz of gelignite and each charge is fired separately.

Coal haulage

The 40-ton coal-hauler units are modified Euclid B.1 chassis, fitted with a fifth wheel and towing 60-yd^3 bottom-dump semi-trailers. Normally, each 'horse' operates with two trailers, one trailer being in transit to or from the screening plant while the other is being filled.

The 40-ton coal haulers are too large for use on public roads and a private coal-haul road, $1\frac{1}{2}$ mile long and crossing a river and two public roads on three Bailey-type bridges, links the site to the screens. The saving in coal-haulage costs, as a result of being able to use these large haulage units, has fully justified the expense of constructing the private road.

Coal screening

The screening plant and sidings are operated by the contractor. The screens incorporate hand-picking belts, blending conveyors and crushers. The products are household coal and blended smalls, the latter passing a $\frac{1}{2}$-in screen, and having up to 11 per cent ash.

Manufacture of explosives

When worked out, the site will have used more than 16 000 000 lb of explosive—mostly AN–FO. This is mixed on site in a central mixing shed using a pan-type mixer in which the iron roller has been replaced with a hard rubber roller. The steel scraper blades of the mixer have been replaced with wood, and the driving motor has been placed outside the mixing shed and connected to the mixer with a long shaft. The fuel oil is coloured so that the mixer operator can readily see when each batch is sufficiently mixed. The mixture is packed in polythene bags 6 in. in diameter for transport to the bore-holes.

Restoration

As work proceeded on the main excavation the topsoil and 3 ft of subsoil were stripped from the area ahead of the other operations. Initially, this material was placed in dumps around the perimeter of the site. Now that the work has been in progress for some years it is possible to lift topsoil and subsoil from the areas ahead of the coaling cuts and spread them directly on to the levelled spoil banks of the de-coaled area. The spoil banks are first carefully graded to designed contours, then the subsoil is spread in two layers 18 in. thick, and finally the topsoil is spread in a 12-in layer. Each layer is rooted and all stones large enough to turn a plough are removed. The work is then inspected before the next layer is begun. Some of the land restored has been in cultivation for two years.

WESTFIELD OPENCAST COAL SITE

The Westfield site is on the Fife–Kinross boundary and has been operated in its present form since 1961. The seams being worked (Figs. 2 and 3) are the Boglochty Series, and in the northeastern part of the site some seams of the Coal Measures are also being worked. The seams outcrop on the south and west, and the basin is cut off by the Ochil Fault in the north. To the east the seams tend to thin out and deepen, limiting development in this direction. Aerial views of the site are given in Figs. 3, 4 and 5 (Plate I).

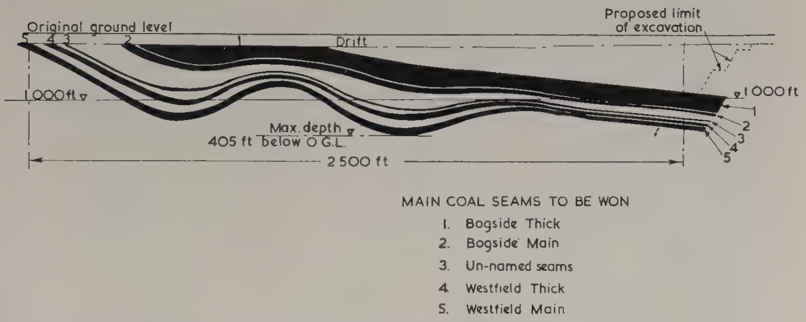


Fig. 2.—Westfield opencast coal site: E–W section through centre-line of pit.

Previous workings

Until 1908 the Westfield shale was worked as a source of crude oil and three small collieries also worked part of the Bogside Thick seam. The seams are very disturbed by folding and faulting and there are

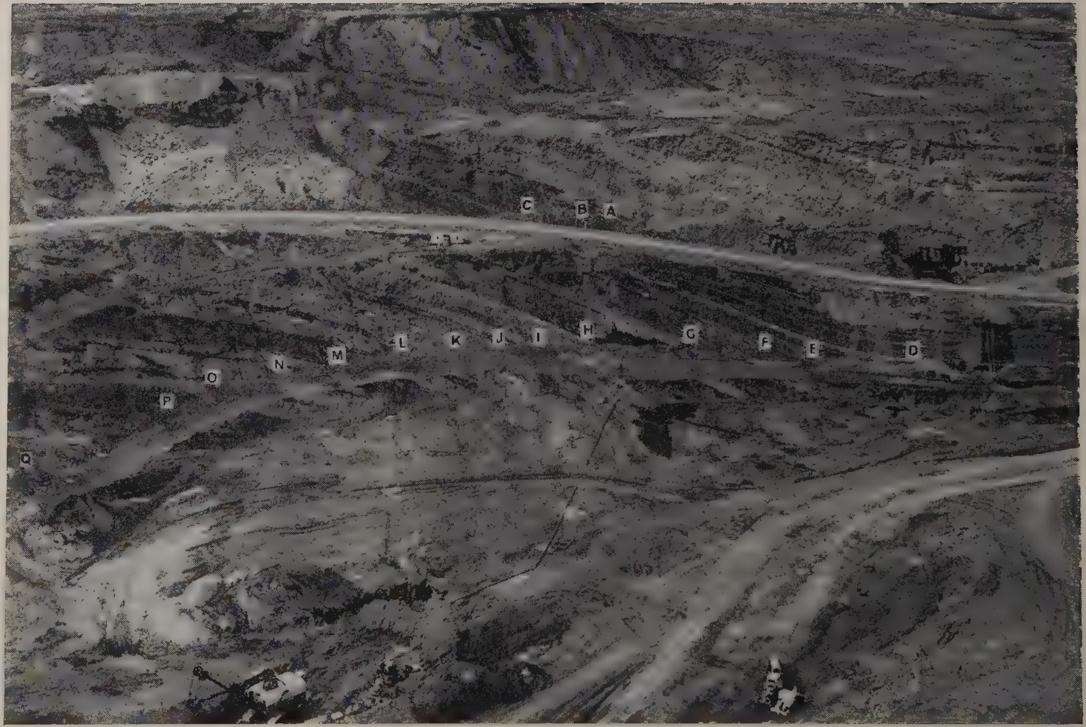


Fig. 3.—Enlargement of area D in Fig. 5. The lettering identifies the leaves of the Boglochty Series.

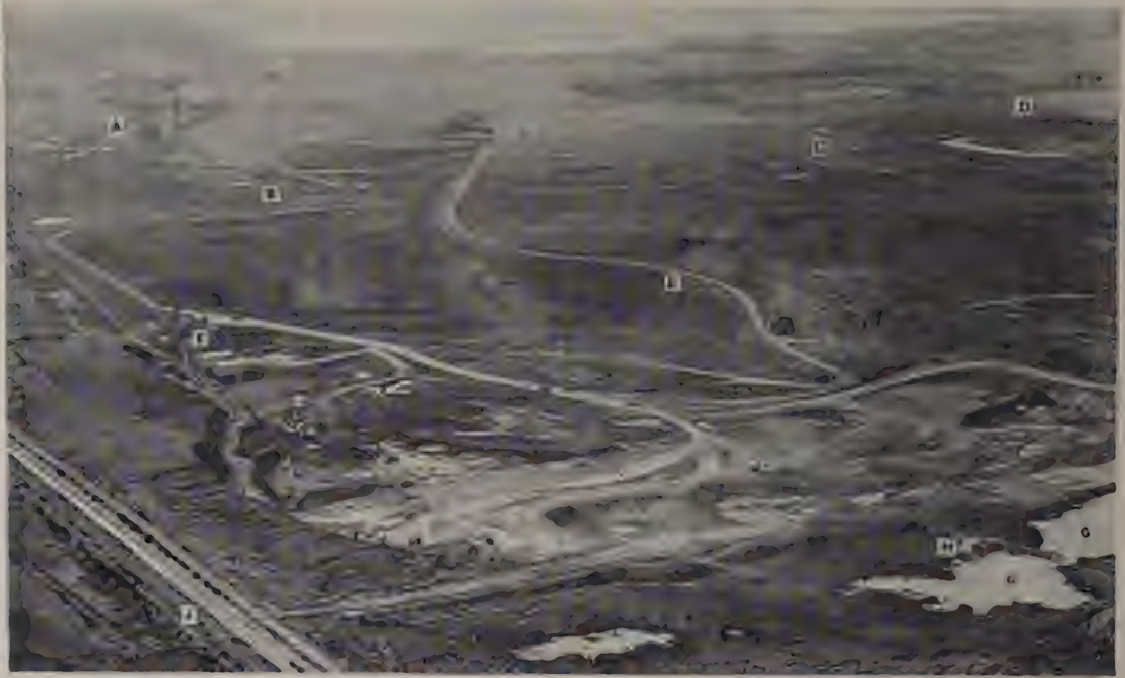


Fig. 4.—Westfield: aerial view of site from southeast, August 1964.
A Lurgi plant; *B* Coal preparation; *C* Workshops; *D* Washery discard and coal slurry beds; *E* Coal-haul road; *F* Crushers; *G* Settling pond for water pumped from pit; *H* Pumphouse for clarified water; *J* Diversion channel for Lochty Burn.

many partings of clay shale and sandstone, making the recovery of coal by normal deep-mining methods difficult and wasteful.

Opening-up the site

Before mining could begin it was necessary to divert a public road and a number of services and watercourses.

The southern part of the site was covered by a peat bog, and since this peat is thixotropic it was necessary to remove it before excavation of the underlying material could begin. It was decided that the most efficient method of removal would be by dredging, pumping the dredged material into lagoons (*E* in Fig. 5). The lagoon walls were constructed from drift material obtained from the northern part of the site, the walls having a maximum height of 65 ft and containing approximately 3 750 000 yd^3. The dredge, of the cutter-suction type with a $7\frac{1}{2}$-ft diameter head mounted on a 71-ft arm, was brought to the site by road, in sections, and built in a dry dock on the site. The dock was then flooded and the dredge cut its way into the area to be dredged. About 4 300 000 yd^3 of peat, sand, silt and clay were removed by the dredge at an average rate of 90 000 yd^3 (solid) per week.

Overburden removal

The coaling area is small (2500 ft by 1000 ft) by comparison with the maximum depth of 400 ft, so it was not possible to work the site by normal strip-mining methods. The first phase of the mine development is, therefore, an open-pit mine, which will serve as a 'box-cut' for the disposal of the overburden from the second phase of the mine development. The overburden and coal partings (26 000 000 yd^3) must be disposed of clear of the excavation area. A suitable disposal area—a valley containing a bog and surrounded by poor agricultural land—was available about a mile south of the site.

Alternative methods of transport for the overburden from the mine to the tip were considered. The route to the tip area crosses a burn, a railway and two roads, and in order to transport the material to the tip in dump trucks it would have been necessary to construct substantial bridges at these points so that haulage could continue in all weathers. The Opencast Executive decided that the more satisfactory method of moving the material was by crushing and conveying. At the time of letting the contract for the first phase of the mine the Executive had installed the conveying plant and part of the crushing plant.

Drilling for blasting

The drilling equipment consists of two Joy 58 BH Champions, one Reich 675, one Reich 600, and one CPTG 800. These are all track drills with the exception of the lorry-mounted Reich 600. The four larger

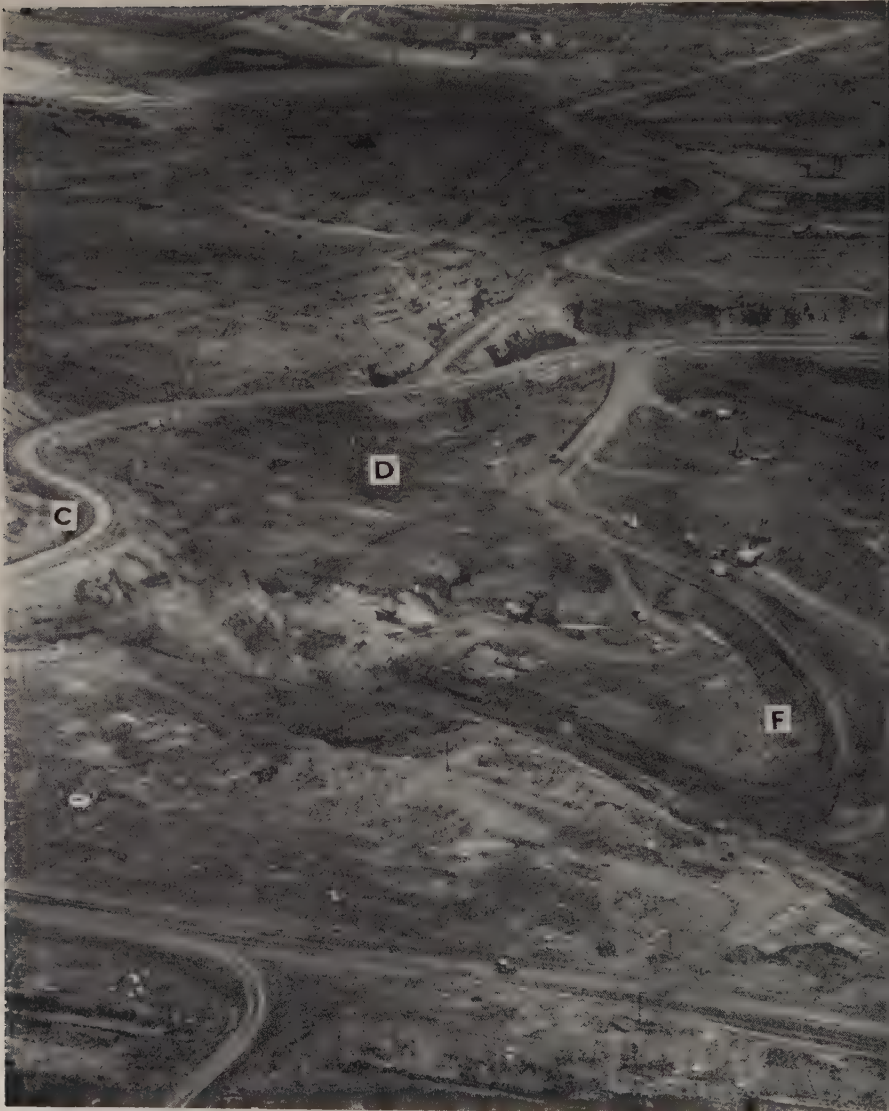
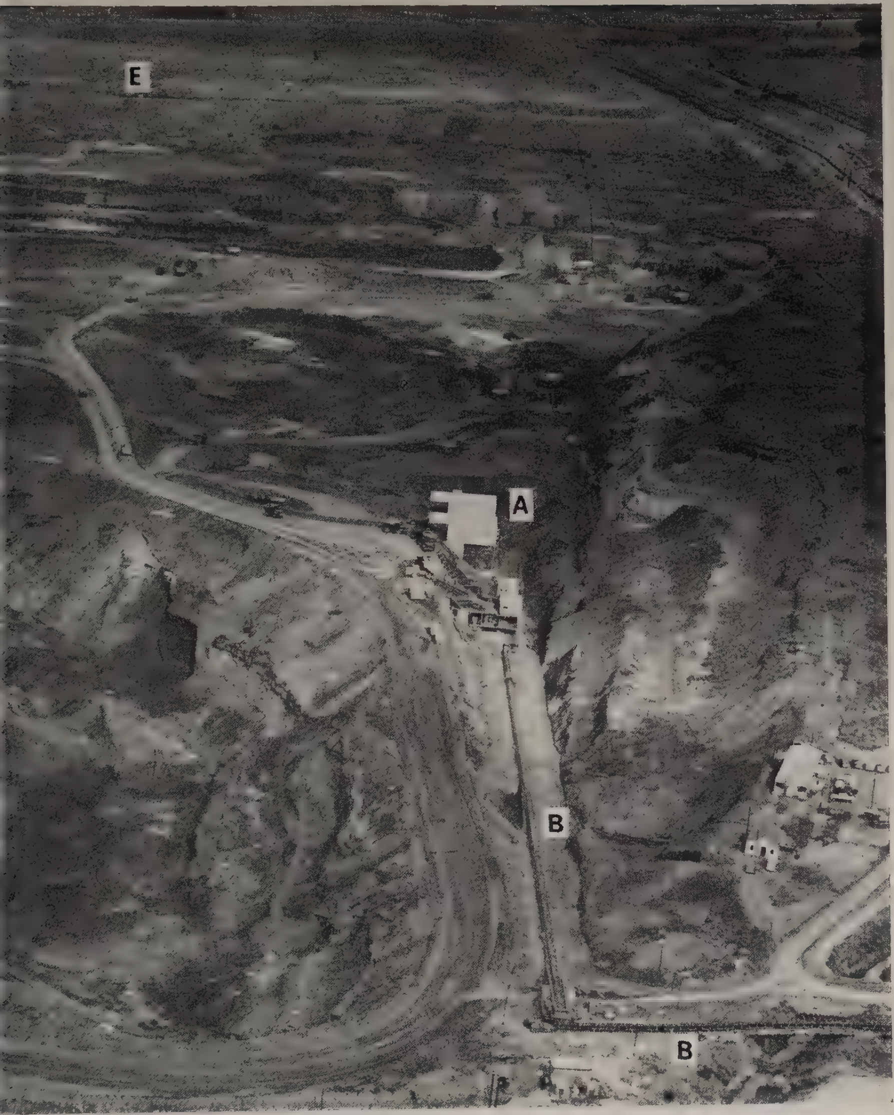


Fig. 5.—Westfield: aerial

A Crushers; *B* Overburden conveyors; *C* Coal-haul road; *D* Exposed st



pit from west, August 1964.
larged on Fig. 3); *E* Peat bunds; *F* Cleaned coal face ready for excavation.

machines use tricone roller bits; the CPTG 800 uses cruciform bits. The two Joy drills and the Reich 675 are used for overburden preparation, drilling holes down to 85 ft for blasting a face 80 ft high. The Reich 600 and CPTG are used mainly for drilling the coal and partings.

Blasting

Initially ammonium nitrate–fuel oil explosive was used at Westfield. Water in the bore-holes caused trouble and it was found that polythene bags did not keep the explosive dry. After unsuccessful experiments with cardboard tubes and rigid polythene containers, it seemed that sealing the explosive in 16-ga. 'tin' cans was likely to be the most successful solution, but there was still some difficulty because the cans tended to float in wet holes and blasting was not uniformly satisfactory.

In conjunction with explosives manufacturers, experiments were carried out using TNT- and nitroglycerine-based slurry explosives which are unaffected by water and very powerful. These explosives are twice as expensive as AN–FO, but they permit the use of larger hole spacings and thus reduce drilling costs.

At first, difficulty was found in initiating the slurry explosive successfully, but a solution was found in using a primer having a very high velocity of detonation. Present practice is to use 5 per cent by weight of primer in each bore-hole, as compared with 10 per cent by weight when using AN–FO.

A factory has now been set up on site to manufacture a slurry explosive using a nitroglycerine base supplied by an explosives manufacturer. The regulations governing the manufacture of this type of explosive on site are stringent, and it is only on a long-term contract that the cost of setting up a suitable factory can be contemplated.

Overburden excavation

Four 150 RB excavators were provided for the overburden excavation: these work across the pit from west to east, removing the overburden in benches 80-ft high. The rock is loaded into 18-yd^3 side and rear dumpers for transport to the crushers. In view of the need to produce rock which can be handled easily by the crushers, primary blasting is much heavier than that required merely for efficient excavation by machines of this size. Notwithstanding this, the excavator drivers have to exercise care to ensure that no rock too large for the crushers is loaded. In the early days of operating the site the experiment was tried of fixing bars across the mouth of the excavator bucket to ensure that oversize stone could not be loaded, but this restriction reduced the output of the excavators so much that it was removed. Instead, a course of instruction, which included a spell of working as a member of the crusher crew, was given to the drivers so that they were clearly aware of the delays and loss of output incurred when oversize rock was loaded.

Coal and partings excavation

The main equipment provided for this excavation consisted of four 54 RB electric face shovels. It was thought at first that it might be necessary to excavate the coal and partings by means of draglines, because in parts of the site the seams stand at a slope of 45°, but after some trials it was decided to excavate the coal and partings in benches 15 ft high, with the excavators rigged as face shovels. At this height the 54 RB can excavate without having to travel up the face.

Since the ash content and sulphur content of the coal vary considerably from leaf to leaf, and indeed from one part of a leaf to another part 50 ft away, it is necessary to expose a variety of coals so that a satisfactory blend of raw coal can be sent to the coal-preparation plant.

The raw coal is loaded into 36-yd^3 rear-dump trucks or 20-yd^3 side-dump trucks and transported to the washery up a coal-haul road of mainly 1-in-12 gradient.

Crushing overburden and partings

Since the overburden and partings contain sandstone, shale, fire clay, etc., it was necessary to provide crushers which could handle this mixed material without clogging. The crusher installation is shown in Fig. 6. The initial equipment provided by the Executive was one double-roll Sheepbridge crusher and one single-roll Sheepbridge crusher, each having an output of 1100 t/hour. The double-roll crusher is powered by two 100-hp electric motors and the single-roll crusher is powered by one 200-hp electric motor. Each crusher is fed from a chute by a heavy-duty apron feeder (25-hp) and a live-roll grizzly (25-hp).

The crushers, which reduce the material to 10 in. or less, are capable of handling rock with a crushing resistance of up to 10 000 lb/in^2 to a maximum size of 24 in. by 30 in. by 54 in. Smaller rocks of up to 15 000 lb/in^2 crushing resistance can also be dealt with.

The drift overlying the rock head contains whinstone boulders which could not be removed during the dredging operation. As these boulders are too hard for roll crushers, two hammer crushers were installed after the start of the contract, driven by 450-hp electric motors and fed by 50-hp heavy-duty plate feeders, to handle this harder rock; each has an output of 550 t/hour.

Conveying and spreading

The crushed rock is fed on to a steel-cord belt 48 in. wide, travelling at 800 ft/min and having a rated capacity of 3440 metric t/hour. The

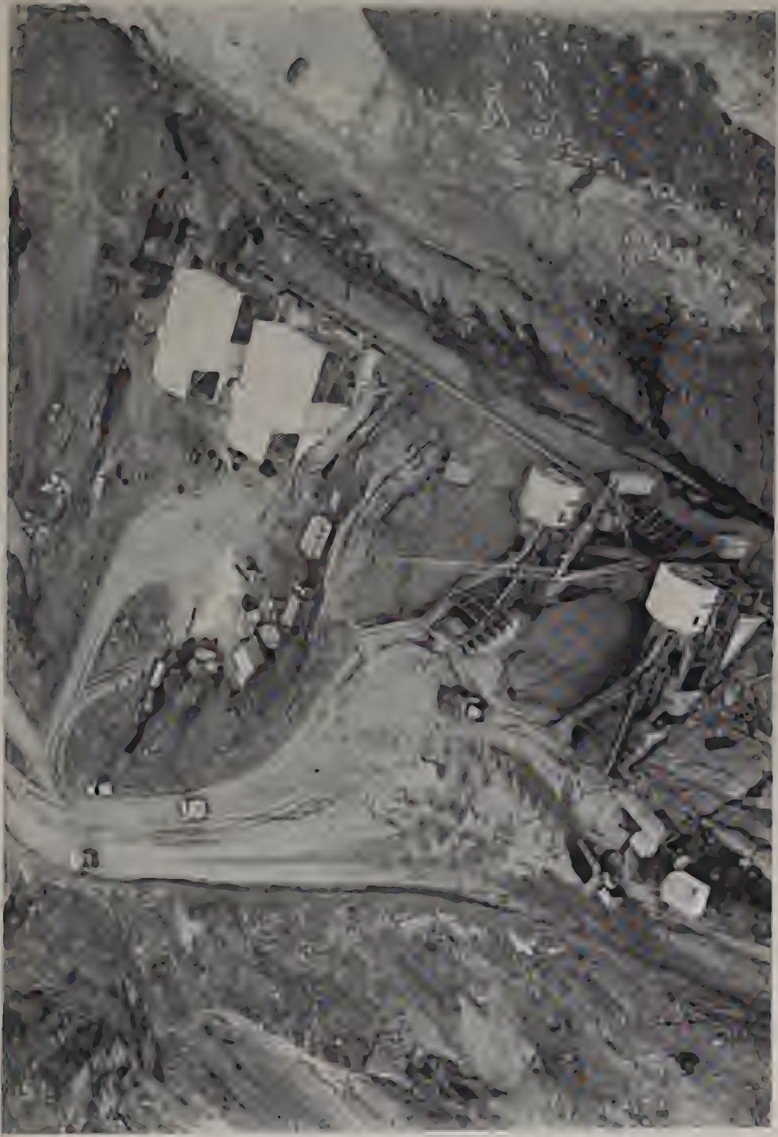


Fig. 6.—Westfield: aerial view of crusher installation.

belt has three fixed sections and one slewing section, the overall length being 11 000 ft. The steel cord has a high-tensile strength and low stretch factor, and is thus suitable for use in long single flights. It also has considerable resistance to impact damage.

The slewing belt referred to above operates on the tip surface, the tail of the slewing belt receiving the material from the head of the last fixed belt and conveying it to the spreader unit. When first installed the slewing belt was extended in line with the last fixed belt. As the material was tipped on the left-hand side of the slewing belt, the belt was gradually slewed in an anti-clockwise direction on the tipped material until it was at right angles to the last fixed belt. The slewing movement having been completed, the slewing belt will now move along the fixed belt and at right angles to it, taking the material off the fixed belt by means of a tripper.

The slewing belt is mounted on sleepers. Flat-bottomed rails, also mounted on the sleepers, run parallel to the axis of the conveyor, one on each side of the belt supports, and serve to carry a tripper unit and the tail of the connecting bridge which takes the material from the tripper to the spreader unit.

The spreader (Fig. 7) is mounted on caterpillar tracks, and has a jib 92 ft long which can be rotated through an angle of 300°. A high-speed conveyor belt mounted on the jib receives the material from the connecting bridge and throws it out to form the tip. The spreader driver sitting in a cabin alongside the spreader jib can control the movement of the spreader, its connecting bridge, and associated tripper, along the length of the slewing conveyor, forming the tip to predetermined contours as he moves. The slewing conveyor is moved by means of a D7 tractor with a special roller attachment, which is first fixed to the flat-bottomed rail on one side of the conveyor; the tractor then moves the length of the conveyor at a slight angle to it, dragging the conveyor towards a new position, a few feet at each pass.

Restoration

As already mentioned, the tip area was a bog surrounded by poor agricultural land. The overburden is being stripped from this area ahead of the tip, and the tip is being formed into a hill having a maximum slope of 1 in 8 and blended into the local scenery. As each section of the tip is completed it is graded and the subsoil and topsoil replaced on top. It will later revert to agricultural use.

Drainage of the mine

Water collecting in the pit is pumped to one of two main sumps by small diesel pumps and then taken by 6-in electric pumps to a settling

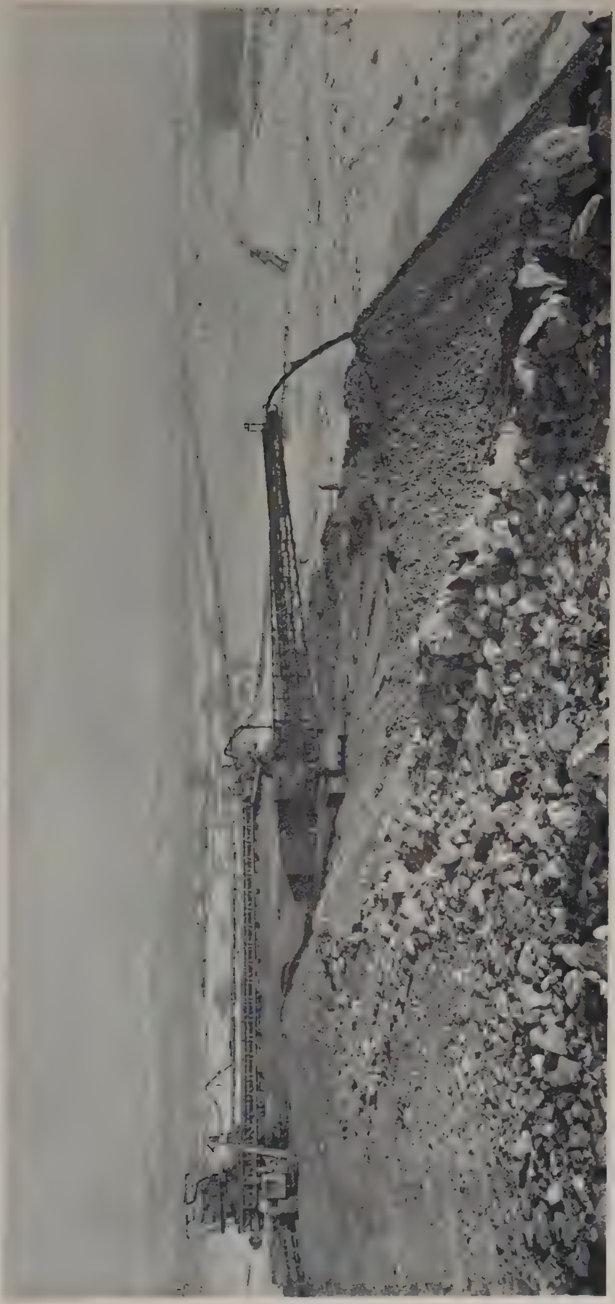


Fig. 7.—Westfield: the spreader installation.

At the extreme left is part of the slewing conveyor, and the specially adapted D7 tractor used for moving it. Behind the tractor is the tripper, which discharges the crushed rock on to the connecting bridge for carriage to the track-mounted spreader.

pond from which the clarified water is pumped into a local stream. Since the drainage water contains much abrasive grit it has been decided not to use high-lift pumps as the mine is deepened, but instead to employ stage pumping to sectional steel tanks installed at intervals of about 120 ft.

Coal preparation

The layout of the coal preparation plant is illustrated in Fig. 8.

The raw coal is crushed to −3 in. and sized to $\pm\frac{3}{8}$ in. The larger material is separated in magnetite dense-medium baths, and the fraction having a specific gravity of more than 1·75 is discarded. The $-\frac{3}{8}$-in material is wet screened, and the $\frac{3}{8}$-in to $\frac{1}{2}$-mm material is separated at 1·6 sp.gr. in a feldspar-type wash-box.

This type of treatment of the raw coal is essential, since the coal seams at Westfield contain many thin partings which could not be removed in any other way.

A novel feature of the preparation plant at Westfield is the method of handling the $-\frac{1}{2}$-mm material. The slurry from the desliming screen is pumped to a thickener via low-pressure hydrocyclones. Originally, this thickened material was transported, by specially adapted dump trucks, to slurry ponds where it was allowed to settle, but now a lock-hopper pumping system enables it to be pumped directly to the slurry ponds.

This lock-hopper system operates as follows. The thickened slurry is pumped into one of two pressure vessels or ‘lock-hoppers’ by means of a low-pressure diaphragm pump. When the container is full the inlet valve is closed automatically and high-pressure clean water is pumped into the top of the lock-hopper. The discharge valve at the bottom is opened and the slurry is forced through the discharge pipe. When the hopper is empty of slurry the discharge valve closes. The high-pressure water-inlet valve is also closed and a fresh charge of slurry is pumped into the hopper, displacing the now contaminated water from the high-pressure pumps. Two hoppers are operated alternately by automatic valves and the system allows the highly abrasive slurry to be pumped at a high pressure without being passed through the high-pressure pumps. The discharge pipe is a 3-in diameter C.I. pipe, 3000 ft long, and the outlet is 80 ft higher than the inlet. The system can handle 150 gal/min with up to 55 per cent solids.

One of the main markets for the coal from this site is a Lurgi gasification plant, which has been built within the site boundary and receives all its coal supplies from the site. The Lurgi plant uses non-coking coal of $1\frac{1}{4}$- to $\frac{1}{2}$-in size only.

All the coal produced at Westfield, including the Lurgi fuel, is handled by rail on private sidings, operated by the contractor.

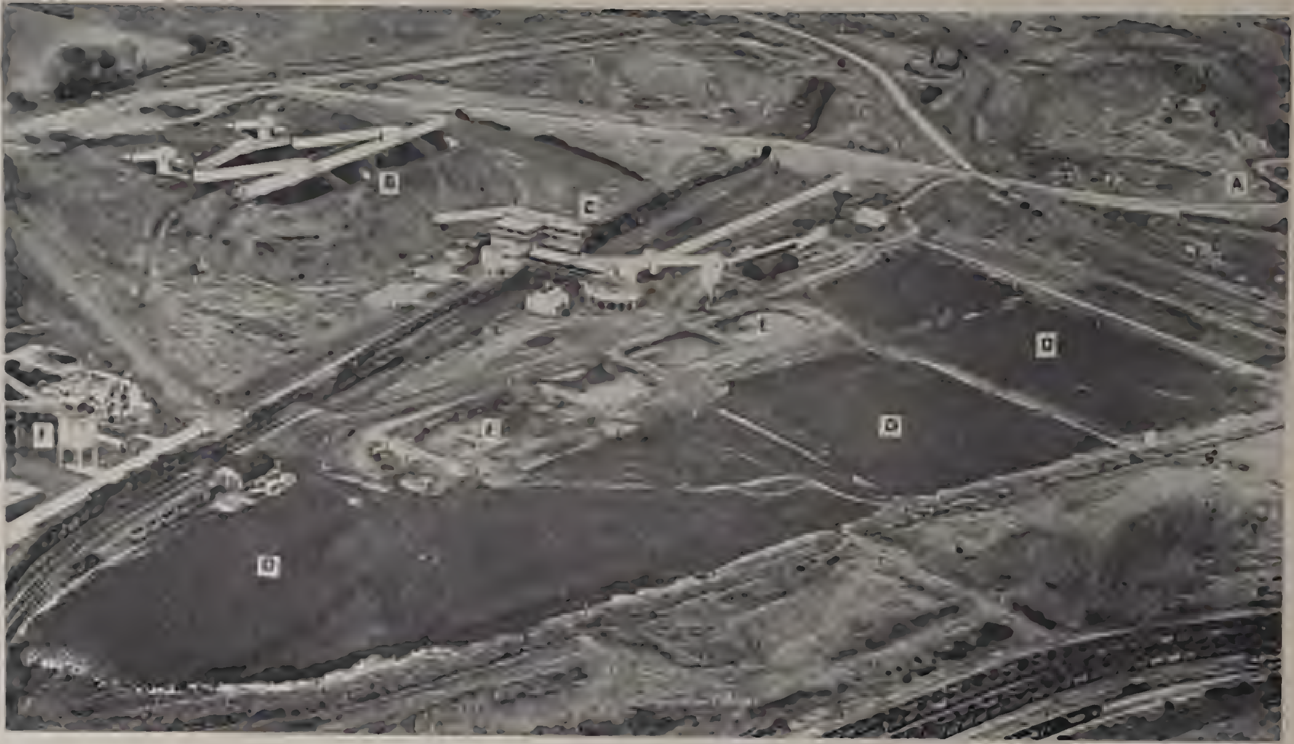


Fig. 8.—Westfield: coal-preparation plant.
A Coal-haul road; *B* Raw coal bunkers; *C* Washery; *D* Washed coal storage; *E* Slurry diversion storage; *F* Coal reception area, Lurgi plant.

GENERAL

Power supply

At both Acorn Bank and Westfield most of the large machines are electrically driven.

At Westfield an 11-kV overhead ring-main has been installed around the pit, and the excavators and pumps are supplied from this via trailing cables from eleven 3·3-kV skid-mounted transformers and field switches. The total installed horsepower is 12 853.

At Acorn Bank the supply to the excavation area is also through overhead lines. Since the two big draglines are American-built and operate at 60 c/s, two frequency-changers have been installed to supply them. The installed horsepower at Acorn Bank is 6000.

Plant maintenance

On each site a large hangar has been erected so that all except the largest machines can be repaired and maintained under cover. These workshops are equipped to carry out major repairs, and large stocks of spares are held. For example, a water-brake dynamometer is installed in each engine shop, and it is normal practice to test-run rebuilt engines before installation.

Site communications

On these large sites communications are vital to ensure complete control of the works without unnecessary delay. For this purpose both sites are equipped with radio and internal telephone systems.

At Westfield, where many operations are concentrated in the pit, the office of the radio controller is sited overlooking the pit. This has proved of great value, for an alert radio controller can sometimes anticipate a possible hold-up in the operation of the plant and take early action to counter it.

Each site has a fleet of Land-Rovers for the speedy movement of personnel. At Westfield the vehicles are equipped with radio and operate as taxis under the orders of the radio controller.

Plant outputs and excavation quantities

The outputs shown in Table I are those actually achieved on the sites. They should not be considered as the maximum achievable by the plant, for there are restrictions which limit the overall output in certain cases. All quantities are in solid measure.

TABLE I.—*Output*

	Drift	*Rock*	*Parting*	*Coal*
1150 B dragline	—	1000 yd^3/h	—	—
150 RB face shovel	450 yd^3/h	350 yd^3/h	275 yd^3	400 tons
54 RB face shovel	—	—	150 yd^3	150 tons

Conveyor system	Approx. 1000 yd^3/h
Roll crushers	,, 500 yd^3/h each
Hammer crusher	,, 300 yd^3/h each

Drills		
Joy 58 BH	65–70 ft/h	$6\frac{5}{8}$ in. dia.
Reich 600	55–60 ft/h	
Reich 675	40–50 ft/h	$6\frac{5}{8}$ in. dia.
CPTG 800	40–50 ft/h	4 in. dia.

Washery	135–145 t clean coal/h
Coal haulage	45–50 t/vehicle/h
Overburden haulage	50–55 yd^3/h

TABLE II.—*Total excavation quantities*

	Overburden *yd^3*	*Coal* *tons*
Acorn Bank	110 000 000	7 000 000
Westfield	25 000 000	9 000 000 (raw) 6 000 000 (washed)

Some cost comparisons

There are some items of equipment which are common to both Westfield and Acorn Bank, i.e. Joy 58 BH drills, 27-ton dump trucks, and Caterpillar D8 tractors. The organization and maintenance set-up is similar and yet some cost items, when compared on a like basis, are so different that they require some comment. Examples are:

(*a*) *Tyre costs on 27-ton dump trucks.*—On both sites, road maintenance is of a high standard, yet the tyre costs at Westfield are 220 per cent greater than at Acorn Bank. Vehicles on both sites are loaded by 5-yd^3 excavators and maintenance procedures are similar. Two differences in site conditions which contribute towards this cost difference are: (1) At Acorn Bank the shale used for surfacing the site roads is comparatively soft and the bulldozers and graders used for forming and maintaining the roads can shear off the inequalities to produce a well graded running surface. At Westfield the rock contains a lot of sandstone which is too hard for the grader and dozer blades to break; consequently, the running surface is always poorer and during wet weather the difference between the running surfaces on the two sites is very noticeable. (2) The maximum gradients on both sites are similar, but at Westfield, owing to greater depth, the length of haul at maximum gradient is greater.

(*b*) *Bit costs on drilling equipment.*—Bits are of similar size and manufacture, and footages per hour are comparable; yet bit cost at Westfield is 73 per cent greater than at Acorn Bank. This can only be attributed to the higher proportion of abrasive materials present in the overburden at Westfield.

(*c*) *Track costs on bulldozers.*—The replacement policy is the same on both sites, but at Westfield the track costs are 35 per cent higher than at Acorn Bank. Once again this can be attributed to the presence of a larger proportion of abrasive materials in the overburden at Westfield.

Methods of payment

Both Acorn Bank and Westfield sites are operated under contract to the National Coal Board Opencast Executive. At Acorn Bank payment is made per ton of clean coal produced. This payment includes all operations—from stripping off the topsoil at the start of the work to replacing the topsoil after the extraction of the coal has been completed and the site levelled.

Normally, no inspectors are employed by the Opencast Executive to supervise the cleaning and loading of the coal at Acorn Bank. Samplers, employed by the Executive, take frequent samples of the coal and coal pavement; these are analysed for ash content and the results of the analysis made available to the contractor. The terms of the contract empower the Executive to refuse to accept coal having an average ash content of more than 10 per cent, and the contractor is required to use the coal-analysis figures to produce a blended product to meet this limit. It is therefore the contractor's responsibility to ensure that the surface of each seam is properly clean and that none of the coal pavement is dug with the coal. Because of the method of payment it is also in the contractor's interest to see that no coal is wasted.

Since, at Acorn Bank, excavation is not paid for as a separate item, the initial excavation of 10 000 000 yd^3 already referred to represents an inordinately large capital lock-up for the contractor, for this work yields comparatively little coal. This item was met by an advance from the National Coal Board which was repaid as coaling proceeded.

At Westfield the normal method of payment has been modified because, while the primary object of the contract is to recover all the clean coal in the basin, the secondary object is to produce a void to receive the overburden from the second phase of the mine. Accordingly, two rates are paid: one rate covers the cost of all excavation plus the disposal of the overburden; the second rate covers the cost of washing and handling the coal, after delivery to the washery.

622.271.3:622.332(43) *Paper 29*

Opencast Technique in German Brown Coal Mining

WIDO TILMANN, Dr.-Ing., Dipl.-Berging., Bergrat a.D.,
Managing Director, Deutscher Braunkohlen-Industrie-Verein e.V., Köln

---

SYNOPSIS

Brown coal in the Federal Republic of Germany, in spite of its relatively low calorific value of 2900–1600 kcal/kg, is the cheapest form of primary energy, owing to its location near the surface enabling it to be worked predominantly by large-scale opencast mining.

Output reaches more than 100 000 000 t/yr to satisfy the high consumption of coal by the brown coal power plants and the demand for brown coal briquettes as household fuel. This output requires removal of more than 200 000 000 m^3 of overburden, which could only be achieved by complete mechanization of the winning and conveying operations.

The bucket-wheel dredge without travelling boom has proved most convenient for winning coal as well as for removing overburden, and has a maximum cutting height of 50 m and a depth of up to 25 m. For transport, conveyor belts (up to 2200 mm wide) and, for greater distances outside the open-cuts, rail haulage with large wagons of up to 96 m^3 capacity are preferred. The overburden (gravel, sand or clay) overlying the coal is dumped on outside dumps or close behind the worked faces or into worked-out fields. This is done by overburden spreaders having a belt-conveyor discharging boom of up to 100 m. Haulage to the dumping area is carried out either by belt conveyor lines or by trains. In the latter case the material is first dumped into a ditch bunker, then taken out by a special scooping machine and led to the spreader by a movable belt conveyor.

The opencasts are becoming deeper and will eventually reach 250 m, necessitating complete dewatering of the opencast area and its surroundings. This underground water drainage affects an area up to 20 km from the opencast boundaries.

Opencast mining severely alters the landscape, involving the removal of agricultural and forestry areas, villages, railways, and roads within the mining region, but restoration and recultivation are practised.

GERMAN BROWN COAL OCCURRENCES AND THEIR ECONOMIC IMPORTANCE

Germany's economic development as an important industrial country has been vitally supported by its coal mining. Besides known coal deposits, particularly in the Ruhr, Germany has vast Tertiary brown coal occurrences, most of them near the surface. The German brown coal mined by opencast methods from early times has, with the mechanization possible in this region, reached a high technical standard of cutting combined with extensive balanced production. An opencast technique has been developed that has come to be regarded as an example to the world.

The total reserves of brown coal in Germany amount to about 100 000 million metric tons, of which more than 60 000 million tons are within the Federal Republic. About 9000 million tons of this amount can be won by present opencast techniques, 85 per cent of it from the Lower Rhine district, where the main deposits lie. Other important occurrences being worked are in Lower Saxony, near Helmstedt, at the Zone boundaries, in Bavaria, near Regensburg, and in Hesse, between Frankfurt and Kassel. Brown coal output has been steadily increased owing to the rising demand for electrical energy, and in the Federal Republic in 1963 amounted to a record 107 000 000 tons. To reach this output 205 000 000 m^3 of overburden had to be removed.

While German brown coal mining had been more or less evenly developed in all districts up to the end of the Second World War, there has been a change in technique since partition. In central Germany lack of raw materials and foreign currency has led to difficulties and has handicapped technical development. In the Federal Republic, on the other hand, excavators and overburden spreaders, as well as means of conveyance, have been newly designed for the condition of the deposits. Mobile earthmoving equipment was taken over from tested American sources and altered to the needs of brown coal mining. In spite of the deterioration of geological conditions, output per man-shift has been considerably increased.

Owing to the moisture content of from 48 to 62 per cent and to the relatively low calorific value of 2900 down to 1600 kcal/kg, brown coal cannot compete with other primary fuels over long distances because the ash content is too high. The market radius of raw brown coal is thus limited, and it was important for the development of the industry to dry and briquette the coal at the site in order to upgrade it and to make it transportable. Today about 40 per cent of the solid fuel used in households in the Federal Republic is as brown coal briquettes.

With technical improvements and the increasing demand for electric power raw brown coal was an obvious fuel for power plants to be erected near the opencast mines. In 1963 the briquetting works took 43 000 000 tons to produce 15 800 000 tons of briquettes, while 53 000 000 tons were delivered to public brown coal power stations, in which about 33 600 million kWh are generated—37·5 per cent of the electricity produced by public concerns.

Coal consumption of the power plants working on brown coal is steadily increasing because existing stations are being expanded and new ones erected.

At the moment there are roughly 6000 MW installed in public brown coal power plants of the Federal Republic, the cheapest electric energy in Germany being produced in these plants near opencasts. While brown coal is responsible for about 12 per cent of the total primary energy production of the Federal Republic, up to 90 per cent of the output in central Germany is based on brown coal.

GEOLOGICAL AND HYDROLOGICAL REQUIREMENTS FOR OPENING UP AND OPERATING BROWN COAL OPENCASTS

Brown coal, in contrast to black coal, is predominantly overlain by loose material such as sand, gravel, clay or loam. The age of the Tertiary brown coal differs, however, in the various occurrences in the Federal Republic. The deposits originated mainly before the great marine transgression of Eocene and Oligocene times, as well as that in the lower Tertiary Miocene and Pliocene. Owing to such differences in age the grade of the brown coals differs within the various occurrences, so that calculated calorific values for substance free of ash and moisture lie between 5700 and 6800 kcal, i.e. 22 500 and 27 500 Btu/kg. From the Lower Rhine, Miocene brown coal has a calorific value of about 1750 kcal/kg (7000 Btu/kg) with moisture content from 60 to 62 per cent and of ash from 2 to 8 per cent.

Faults mainly striking southeast to northwest, caused by bulges in underlying rock formations, have split seams originally horizontal varying in thickness from 10 to 60 m, and averaging 40 m.

The most important fault is the ‘Erftsprung’ with its side effects. In this area the seam is partly lowered to 600 m, mining mainly taking place on the high Villescholle, which expands on the western side of Cologne to about 40 km long from south to north and 3–4 km wide.

After those parts with a good overburden ratio had been mainly exhausted, the question arose—how to deepen the open-cuts. Investigations led to the conclusion that work might be continued economically under the conditions described down to a depth of 250 m.

The new open-cuts show much thicker layers of overburden as well as more widely spread tectonic disturbances and have led to a change in the structure of the brown coal mining industry in Western Germany. The coal seam is often split by interbedded seams of clay and sands, so that it cannot be won on one bench. Thus the necessity arose of taking out both overburden and coal on the deeper benches. While in the old mining districts the ratio of overburden to coal is about 0·35 : 1 up to 0·5 : 1, it worsens distinctly in the new fields and will frequently be about 3·5 : 1—in less favourable fields even up to 6 : 1.

To permit work in open-cuts complete dewatering of the overburden and coal is necessary, since it is essential to take the artesian pressure off in the base of the seam. At the same time the underground water level must be lowered in the neighbourhood of deep open-cuts to guarantee safety. Dumps also must be dewatered and action must be taken to prevent surface water from entering the workings. A systematic and planned drainage of the rainwater in the whole area of the open-cut has thus to be carried out.

In the Lower Rhine area dewatering is carried out by deep wells because the lowest benches of the deep workings will be down to 250 m underneath the original surface. In addition, in the Erft Valley to the west there are diluvial and Tertiary gravels and sands of great porosity which alternate with layers of clay, and there the underground water-level must be reduced to 250–300 m, a lowering which has affected a wide area around the cuts extending up to 20 km in length. In the Erft Valley approximately 9000 to 10 000 million m^3 of residual water, not including the daily make, has to be removed. Before pumping was begun the continued water supply for industrial, agricultural and domestic use was ensured.

OPENCAST MACHINERY

An important prerequisite for deep open-cuts is that there should be large closely spaced coalfields of the necessary guaranteed life to make it possible to work with big excavators and overburden spreaders and corresponding means of conveyance. Production must not be limited by villages, roads, railways or rivers, the removal of which is technically possible and economically reasonable, but must be determined by the location of the deposits. To concentrate production it is essential to develop big opencast units. The mine-owned railway connects the large opencasts of the Ville and the northern district of the Rhine, and their consumers.

Fig. 1 is a map of the Lower Rhine District, and Fig. 11 (Plate I) is an aerial photograph of the Fortuna opencast mine, showing the extent of the operation.

Brown coal mining is primarily a problem of moving materials and thus current opencast techniques embrace bigger and bigger excavating, conveying and dumping machinery, as well as a steady reduction of personnel in relation to increased output. For example, in the Fortuna open-cut more than 110 units of material (t of coal and m^3 of overburden) are handled per man-shift or, in terms of coal production, 46 t per man-shift.

Complete mechanization of opencast production was attained long ago, but improvements will go on within the range of technical development. Modern communications and remote-control facilities are used to control the widespread working sites. At the same time automation has been introduced and has already shown promising new ways of excavating, conveying and dumping. Only in this way can the increase in labour costs be halted and full profitability be kept up.

An important condition for success in present opencast techniques is

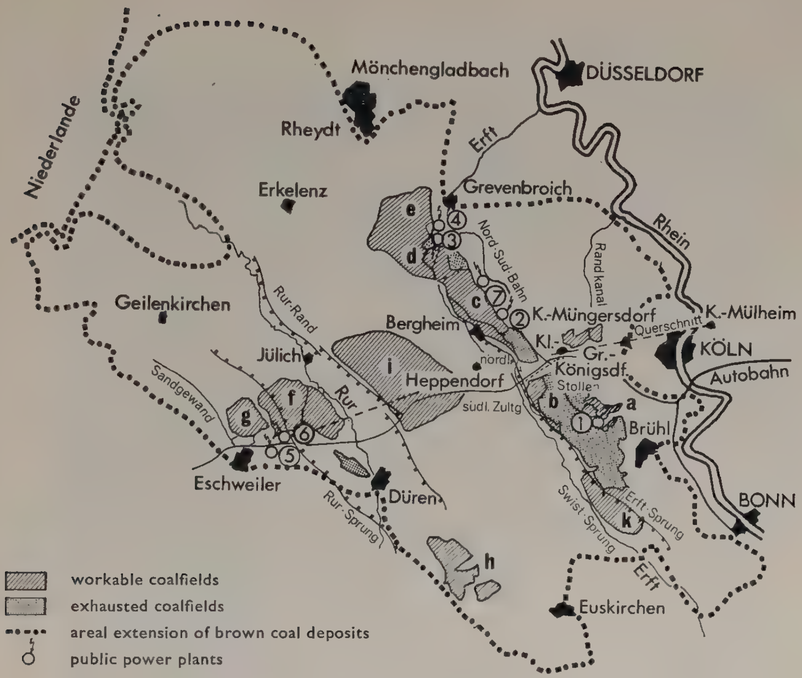


Fig. 1.—Map of the Lower Rhine brown coal district.

OPENCASTS
a Vereinigte Ville
b Frechen
c Fortuna
d Frimmersdorf-Süd
e Frimmersdorf-West
f Inden
g Zukunft-West
h Victor

POWER PLANTS
1 Goldenbergwerk
2 Fortuna
3 Frimmersdorf I
4 Frimmersdorf II
5 Weisweiler II
6 Weisweiler I
7 Niederaussem

FUTURE FIELDS
i Hambach
k Donatus

full utilization of machinery, which must not be interrupted. Work of secondary importance which does not fit the size of the big main machinery must be done by suitable subsidiary equipment.

Winning equipment

The introduction of excavators to enable continuous winning, made possible by the composition of the overburden as well as the low hardness of the coal, led to the introduction of bucket chain dredges. Such dredges, mounted on rails or caterpillars and built as down-dredges or turret dredges with a total cutting depth of 40 m, service weights up to 1400 t, bucket capacities up to 2·24 m^3 and an output of up to 50 000 m^3/day, have been constructed and are still in use. These machines do additional heaving work when used for deep cutting. The main element of their construction is the bucket ladder carrying the

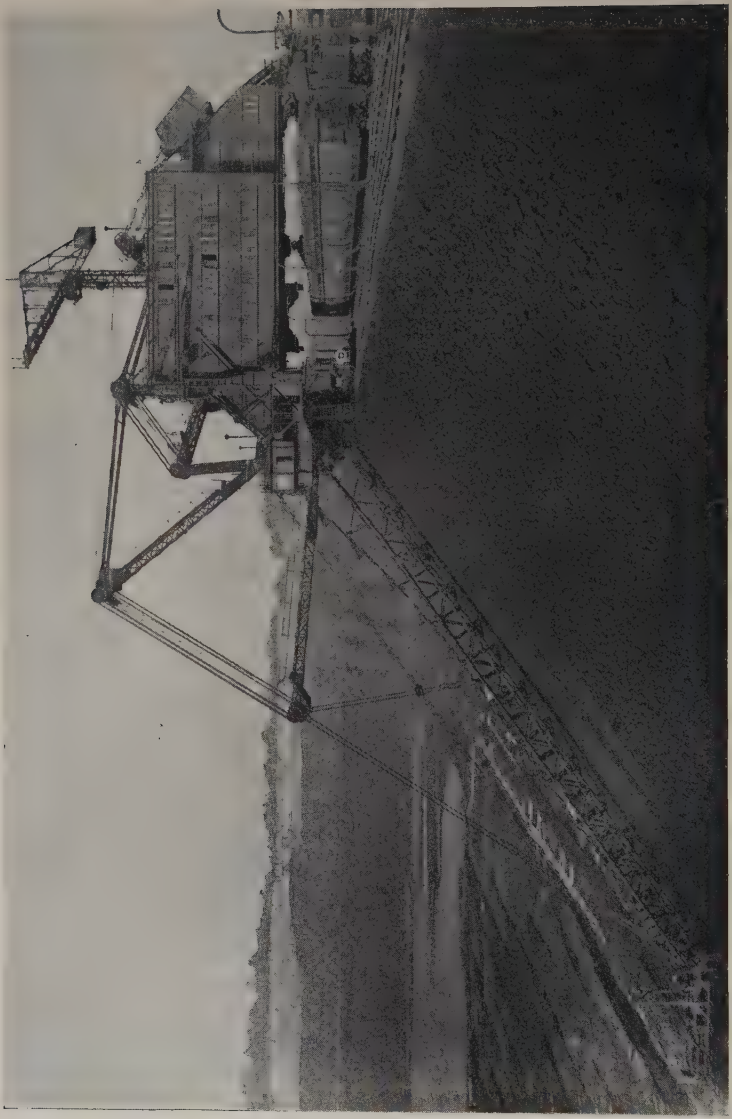


Fig. 2.—Chain bucket excavator.

bucket chain. This is suspended from a boom by means of wire ropes and is brought into the necessary position by winches. The bucket chain consists of two equally long endless link chains, to which the buckets are fastened by rivets at the fourth or sixth link. A bucket at every fourth link leads to a higher number of discharges and a correspondingly higher output, but for working on heavy soil a sextuple linking is chosen for perfect emptying.

The bucket chain moves on rollers on the top side of the ladder and on wearing strips underneath. The chain moves over the tumbler sprocket in the dredge house and over the bottom tumbler at the end of the bucket ladder.

Buckets, either riveted or welded, are provided with highly wear-resistant lip blades made of special grades of steel. Depending on the digging resistance, the blade has sometimes to be provided with additional teeth. The bucket shape must be of such a kind as to facilitate complete and quick emptying. From the chute into which the buckets empty, the material is fed on to a conveyor, the capacity of the transport system having to meet full output. Big dredges necessitate devices for controlling the relative speed of train and dredge.

Generally speaking, a bucket chain dredge cannot excavate different layers separately; the bucket chain is, owing to its method of operation, exposed to heavy wear also, and these are substantial disadvantages which do not apply to bucket-wheel dredges. Thus, owing to the need to open up new deposits in unfavourable conditions, the bucket-wheel dredge has quickly become the dominant type of excavator. They have been designed to all sizes and variety of applications, both for the main excavating (e.g. high-capacity machines) and for auxiliary purposes or preparation work.

The principal difference in these continuous-excavating dredges lies in the different tasks set their actual digging parts, i.e. the buckets. Whereas with the bucket chain dredge they have to loosen the ground as well as lift it up to the working level, with the bucket-wheel dredge these two actions are carried out by two different devices. The buckets of a wheel dredge serve only as a digging element, and all further handling of the dug material is carried out by the succeeding belt conveyors. From this separation of the duties derive many of the advantages of the bucket-wheel dredge, the most essential being the following.

(1) The output can be increased to a certain extent, since the circumferential speed of the bucket wheel is adjustable to the digging resistance of the soil, and can vary from 1 to 4·6 m/sec, which means a cutting speed four times as high as that of a bucket ladder. Thus 90 discharges per minute are obtainable.

(2) Separate digging of coal and overburden is possible, no matter where they are located at the face.

(3) Power consumption and wear are less than those of a bucket chain since the bucket wheel serves only as an element for loosening material, whereas the bucket chain, already exposed to heavy wear, has also to convey the material up to the discharging chute of the dredge.

(4) The digging methods of bucket-wheel dredges are superior.

Arising out of the technical and economic experience gained during ten years of operation with big bucket-wheel dredges, this type of machine is now the most favoured for operation in the brown coal mining industry.

With regard to design features bucket-wheel dredges can be divided into three main groups:

(1) high cut dredges with crowd action, i.e. sliding booms travelling forward and backward;
(2) high dredges without crowd which have to be moved for cutting new slices; and
(3) combined high and deep dredges without crowd.

For large machines the type without crowd action has been adopted after thorough investigation. Such a machine comprises a digging unit, a separately movable discharging unit, and a telescope-type intermediate conveyor bridge connecting the digging and the discharging units. The advantages of these machines are: (*a*) saving of weight; (*b*) simpler operation; and (*c*) easily achievable automatic control of the digging process—of great value for obtaining steady performance.

The most important part of a bucket-wheel dredge is the bucket wheel. At its circumference there are 6 to 12 buckets. The speed of wheel is variable; it is located at the end of a long and slewable boom which can be raised and lowered. The buckets are similar in design to those of a bucket chain dredge: they empty at the upper part of the bucket wheel where the spoil is taken over by separate conveyor. The most modern wheel type, the cell-less bucket wheel, is followed either by a rotating disk, a set of big rollers, or a short belt, all of which throw out the material on to the main boom belt conveyor. From this conveyor the material is fed to the next flight of conveyor, the tail pulley of which is situated at the slewing centre-line of the upper carriage. Depending on the design and the size of the machine this flight (sometimes two) transfers the material to a longer conveyor belonging to a separately slewable discharging boom or to an intermediate conveyor bridge leading to a separately movable discharging unit mounted on caterpillars.

It is typical of all bucket-wheel dredges that the transport of the material is carried out by belt conveyors. Mostly they have troughing idler sets consisting of three high-duty rollers and for the high-capacity dredges the belt widths are up to 2·2 m and the speeds reach 4 m/sec. At the transfer points there are considerable falls and measures have to be taken to prevent belt wear as well as to avoid blockages of the transfer chutes. On the whole, the requirements are similar to those of the large, long and movable or stationary conveyor lines leading from the dredges to other plants. The development of such large conveyor systems has a direct influence on the design of belt conveyors on the large machines.

Where there is no need to dig the different materials separately, the block method is frequently used (Fig. 3). Then the machine travels ahead and digs at a small face across the direction of the long face, the maximum block width depending on the length of the bucket-wheel boom—up to 100 m.

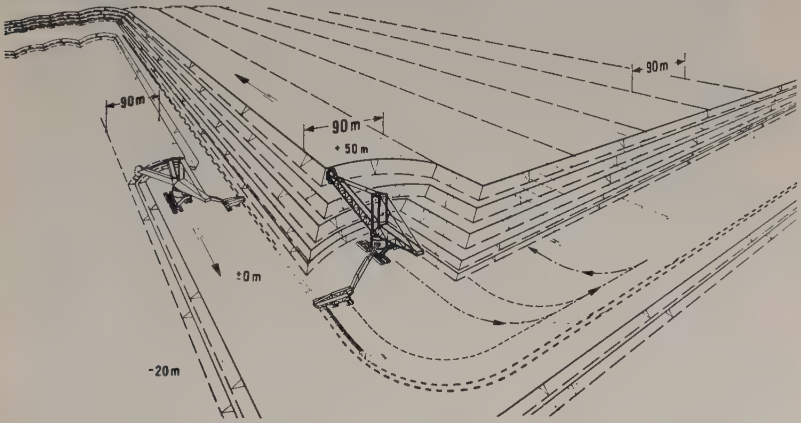


Fig. 3.—Diagram illustrating the working sections for the block method using a wheel excavator.

The faces ahead and abreast of the machine have to have a certain gradient for stability reasons. Such slopes are obtained by travelling into different positions and altering the slewing angle when the bucket wheel is changed from one cutting level to another. At all cutting levels the slices taken are greatest directly ahead of the machine and thinnest at the ends of the cuts. For constant ouput in spite of the varying slices it is necessary to regulate the angular velocity automatically corresponding to the slewing position. Slewing direction reverse and crowd are also automatic.

With a machine without crowd action the movements of the bucket-wheel boom are always preceded by movements of the caterpillars, so that its travel is five times that of a machine with crowd. This affects wear of the travelling gear as well as the working level, which must allow a higher ground pressure.

During the last fifteen years more than 200 bucket-wheel dredges of various sizes and a total weight of 150 000 t have been constructed in the Federal Republic, and nearly half of them were commissioned by foreign companies for use in brown coal or other mineral open-cuts and in huge earthmoving enterprises. As most of the machines have been specially designed for their particular application there are many different types, the biggest being at work in the German brown coal mining industry. Some particulars of one of those working in the Fortuna open-cut may give some idea of the great size of such machines (Fig. 4).

The working height is 50 m and the cutting depth below level 25 m. The cell-less bucket wheel, 17·5 m in diameter, has ten buckets, each with a capacity of 3·8 m^3. The nominal number of discharges per minute is 38, resulting in a theoretical output of 8700 m^3/h. The maximum reach of the bucket-wheel boom is 71·5 m, and the distance between the slewing centre-line of the excavator and the centre-line of the tail pulley of the discharging unit is 130 m. The conveyor belts have a velocity of 3·8 m/sec. There is a main conveyor flight and six auxiliary ones. The service weight of 7400 t is distributed over a large bearing area of three crawler groups each consisting of two twin caterpillars. The machine thus creates a ground pressure of 1·4 kg/cm^2. At the top of one of its 70-m high columns there is an anemometer which will switch off the main drives of the machine if the wind velocity exceeds a certain limit. The total electric load amounts to 15 850 kW, of which 1760 kW is needed for the bucket-wheel drive.

Equipment for overburden disposal

The coal is conveyed to the power stations and briquette factories, but the overburden has to be transported to dumps within the exhausted areas or, in the initial stages of operation, to outside dumps. If transport is exclusively carried out by belt conveyors, as is usually the case with inside dumping, the spoil will be fed directly on to the overburden spreaders. However, if there is a dredge-train operation for long hauling distances, the material is tipped from the trucks into a trench where it will be reclaimed by a ditch bunker loader (Fig. 5) connected to the stacker by a belt-conveyor system. With small plants the scooping device is part of the spreader.

The method of operation necessitates equal capacities on the dredge and stacker side. Adequate stacker performances demand constructional improvements and finding a favourable spot for a carefully designed trench for dumping. To this end separate scooping units are employed to establish a safe ditch bunker. Only in this way can the requirements of a reliable tipping and scooping operation be met, i.e. (1) a stable bunker slope; (2) sufficient clearance between the scooping

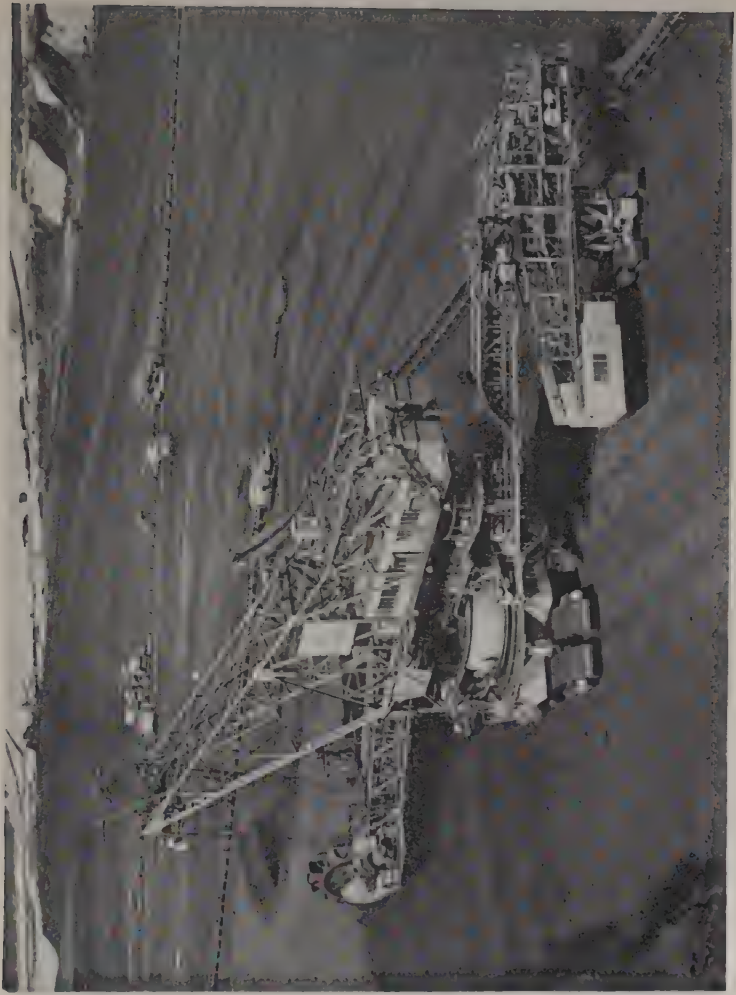


Fig. 4.—Bucket-wheel dredge.

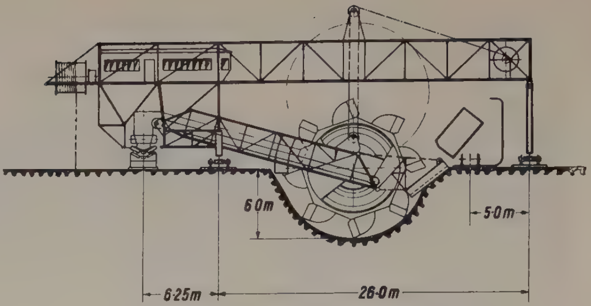


Fig. 5.—Diagram of ditch bunker loader.

device and tipped trucks; (3) uniform deposition of overburden within the trench; and (4) adapting trench volume to the truck capacity.

Troubles in dumping areas caused on the one hand by the extremely high wheel loads of the big overburden trucks and on the other by non-stationary tipping trenches could be eliminated by changing from rail transport to belt conveyors within the actual dumping fields.

The scooping device can be either a short bucket chain or a bucket wheel, but on modern large plants the bucket wheel is usual. The whole machinery is mounted on rails located on both sides of the open-cut. The main structural framework is of bridge shape, enabling the bucket wheel to be raised and lowered easily. Such a design is to be preferred to a cantilever-type machine which would mean waste of structural material. One of the various reclaimers has a service weight of roughly 600 t; the wheel load of the travelling gear is 20 t. The wheel with seven buckets, each of a capacity of 5·0 m^3, has a diameter of 12 m. It is driven by two 430 kW motors. The output amounts to 8500 m^3/h, and the total electric load 1500 kW.

From the ditch loader the material is transported to the disposal unit via a movable belt conveyor, from which its transfer to the first conveyor flight on the disposal unit can take place at any point of the conveyor as it is fitted with a tripper, i.e. a travelling 'belt loop' having also a short slewable crane belt. With larger plants the tripper is frequently mounted on caterpillars rather than rails.

The spreader (Fig. 6), i.e. the disposal unit, mainly comprises three sections: the undercarriage, the slewable middle part carrying the first conveyor flight connected to the tripper's crane belt, and finally the superstructure containing the slewable long discharging boom and the counterweight. The service weight amounts to 2400 t. The travelling gear consists of three twin crawlers having a bearing area of such an extent that a mean ground pressure of about 1 kg/cm^2 will not be exceeded. Since there are two steerable caterpillar groups, the machine has a radius of turn of about 75 m. All the crawlers have individual drives with planetary gears, their travelling rate being up to 6 m/min.

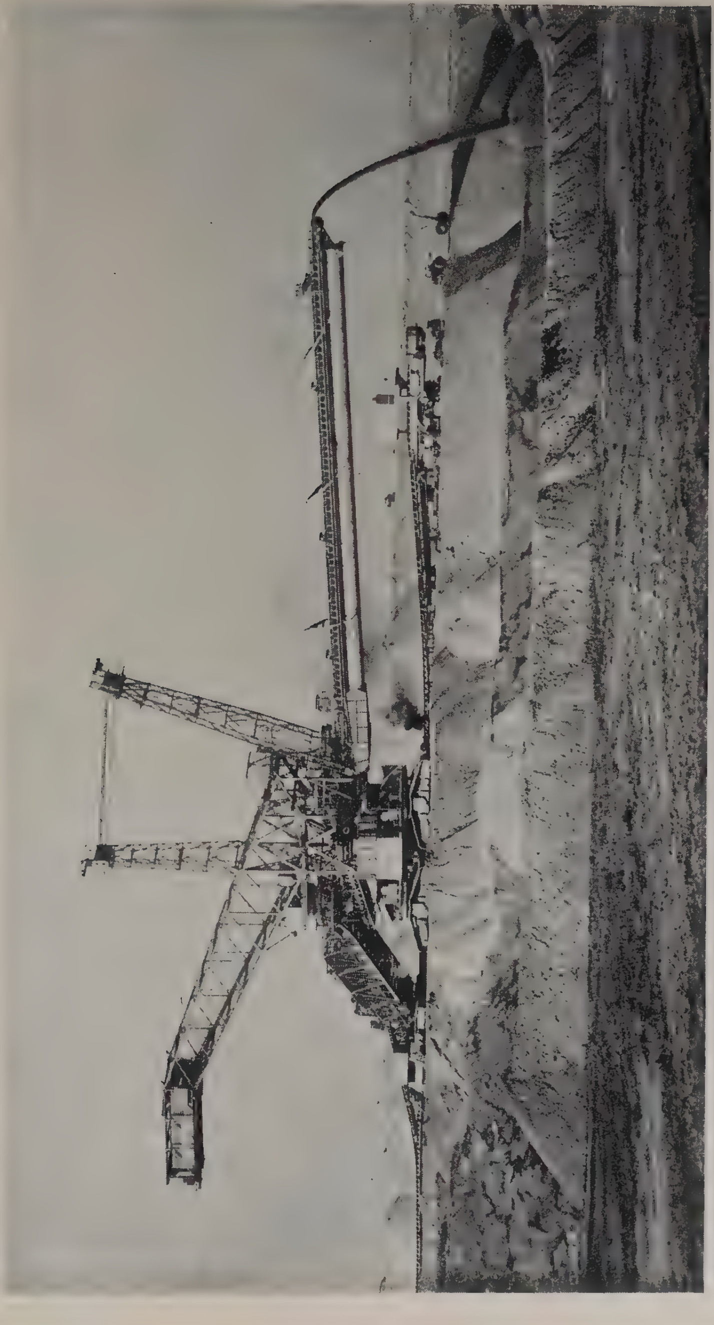


Fig. 6.—Overburden spreader.

Between the undercarriage and the middle section there is a turntable race 15 m in diameter, enabling the middle and upper parts to be moved through an angle of 360°. The respective gears are located in the undercarriage.

All the intermediate conveyor flights connecting the taking-over conveyor with the discharging conveyor are mounted on the middle part. The first conveyor can be designed as a telescope-type bridge effecting freer movements of the entire plant.

The superstructure with its discharging boom and counterweight boom, as well as its two lattice towers supporting the hoisting rope pulleys, slews on the middle section through an arc of 105° on either side of the neutral position. The total height of the machine is 55 m at the towers. The discharge boom itself is either a lattice girder or a thick tube suspended from the first tower. Its length from the slewing centre-line to the discharging drum of the conveyors reaches 100 m. The belt runs at a rate of up to 7 m/sec, which means a useful extension of the throwing range. Such an extended range is of special importance when operating a deep dump since the material should first be stacked as far from the foot of the dump as possible. The length of large machines can reach 160 m. On the counterweight boom there are, as well as ballast, the hoisting winches and the basic electrical equipment. The total electric load amounts to more than 3800 kW.

Depending on the scoop unit, the stacker is designed for spreading 8500 m^3/h of loose material. The discharge boom, having a maximum inclination of 20°, allows high dumping of up to 35 m. For deep dumping the boom will be lowered into the horizontal position. The connecting belt system will only need to be shifted after dumping a block 80 m wide.

Means of conveyance

Developments in means of conveyance have always been strongly linked with those for excavating and disposal machinery. Nowadays either large railway haulage using 900 mm or standard gauge or belt conveyors 800–2600 mm wide are operating, and a combination of both types of conveyance will, as already outlined, be chosen if long distances outside the opencut are to be covered.

In the Rhine District the large wagons running since 1954 have proved satisfactory. For coal haulage 120-m^3 saddle-bottom units with a carrying capacity of 90 t are common (Fig. 7). Overburden is transported in 96-m^3 wagons carrying 180 t (Fig. 8). The coal wagons are mounted on four axles and the overburden wagons on eight, resulting in both cases in an axle load of 30 t. Such high loads required the development of a special heavy rail weighing 64 kg/m.

Trains usually consist of eight wagons and are generally pulled by

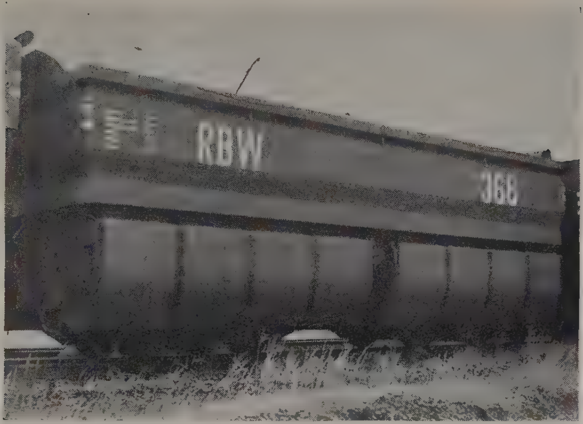


Fig. 7.—90-ton coal wagon with side discharge doors.

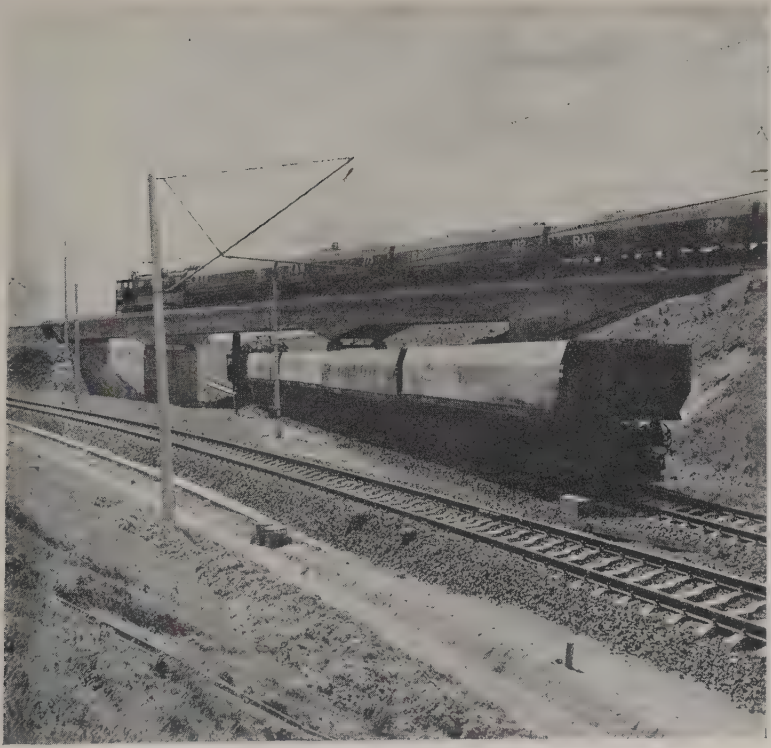


Fig. 8.—Side-tipping wagon for overburden, 180-t capacity.

120-t electric locomotives. Only in rare cases are diesel engines used. The electric machines of standard gauge are mostly converter or rectifier locomotives fed by one-phase a.c. of 50 cycles.

Handling of material by belt conveyors has greatly increased in recent years, utilization having been made possible by solving the problem of shifting long conveyor systems by heavy tractors. In the open-cuts of the Federal Republic there are belt-conveyor lines with a total length of 120 km.

The following are features of present belt-conveyor technology:

(1) Capacity can amount to 15 000 m^3/h; this means belt widths of up to 2200 mm and velocities of up to 6·5 m/sec.

(2) The driving power of one flight goes up to 3600 kW; the power consumption per cubic metre of handled material and per kilometre hauling distance is, for the time being, roughly three times as much as the corresponding figure for standard-gauge rail haulage.

(3) To keep the number of transfer points as low as possible the length of the single flight is increased and the longest has now reached 1800 m.

(4) Large belt systems are provided with steel cord belts of the German Standard types 'St 2000' or 'St 3150', the values being the ultimate strength in kg per centimetre belt width. Thus with belts 2200 mm wide the total permissible traction will be 44 t and 63 t respectively, if a safety factor of 10 is taken into consideration.

The main advantage of belt transport lies in the fact that such conveyors can be run on comparatively steep slopes. Whereas with trains gradients of 1 in 50 are rarely exceeded, inclinations of 1 in 3 are commonly possible with belt conveyors. Installing conveyors instead of railway haulage therefore avoids using large areas which would otherwise be needed for long tracks leading in wide curves up to the surface. That is especially true for deep opencasts where inside dumping would not only be delayed by rail operation, but also rendered more difficult. Belt transport increases the effectiveness of the opencast machinery remarkably, as it allows a continuous flow of material. Interruptions of operations caused by changing trains, derailments or even shortage of rolling stock are eliminated. Capital cost as well as power requirements are higher for belt conveyors than for railway operation, so rail haulage remains important when long distances have to be covered on horizontal surfaces, e.g. connecting the dredge levels with the outside dumps located far off the site. For the time being, of the total transportation capacity rail haulage accounts for 82 per cent and belt conveying 17 per cent, the remaining 1 per cent being carried out by mobile earthmoving equipment.

Auxiliary Equipment

Important for achieving the required output from a large machine is undisturbed operation, i.e. besides keeping it maintained, all subsidiary earthmoving must be carried out by smaller specially designed units, with the consequence that rapid development of auxiliary machinery has taken place; any manual work on this field is out of the question.

At first the operation of auxiliary equipment was largely influenced by American experience with mobile earthmoving equipment and, not surprisingly, after the war the first machines purchased were mounted on giant tyres instead of crawlers. More recently, however, crawler-mounted machines have become more important for special applications.

The advantages of pneumatic tyres as compared with crawlers are: smaller turning radius, larger economic range of application, and higher speed, with travelling permitted on public roads. In periods of bad weather, tyre-mounted machines are, however, at a disadvantage on account of smaller bearing areas, especially when having to travel on unconsolidated paths for long spells; their efficiency drops as they damage roads and risk getting stuck in the mud. Maintaining all roads and levels in a comparatively good condition is, however, of vital importance for all opencast activities.

The crawler-mounted tractor has the advantage when unfavourable ground and weather conditions prevail. The total weight as well as the propulsive power is distributed over a considerably larger bearing area, so slipping rarely occurs, and pushing force and climbing ability are unimpaired.

The importance of auxiliary equipment for brown coal opencast operations is emphasized by the number of machines utilized—over 300.

The development of auxiliary equipment actually started when the first Tournadozer was put into operation in 1952; opencast mining today is hardly conceivable without tractors and dozers. They are used for nearly every kind of subsidiary work and, therefore, with a propulsive output of up to 320 hp, are equipped with various devices, e.g. different blades, buffer blocks, scarifiers, winches, several loading devices, cranes, etc. Of major importance are the tractors fitted with cranes, which became necessary for moving belt conveyors on dredging and dumping levels.

Besides the types of mobile earthmoving equipment mentioned, motor and trailer scrapers mounted either on crawlers or tyres, graders, shovels, trenchers, movable or stationary cranes, truck-cleaning machinery, drilling rigs and track-maintenance equipment are also needed.

Owing to the rough conditions in opencast mining, mobile earthmoving equipment is severely tested. The entire machine must therefore not only be exceptionally sturdy and wear-resistant, but at the same time high-powered and economical in performance. Furthermore, it has to meet all the numerous safety requirements and working conditions.

DEWATERING

The necessary lowering of underground water is done by deep filterwells with submersible pumps of up to 1000 kW. With eleven stages these pumps lift 15 m³ water per minute a height of 300 m; they weigh about 10 t and are about 9·2 m long. They are designed for installation in a filterwell of 800 mm minimum diameter. Wells are sunk with deep boring facilities, using airlifts, the boring cut diameter being 2 m. The greatest depth reached so far is 500 m. In the Fortuna open-cut more than 200 wells have been sunk, of which about 180 are operating constantly. In the whole Lower Rhine District about 550 wells are at work. The underground water-level must be kept about 50 m beneath the deepest point of the open-cut.

In 1955 about 150 000 000 m³ of water were pumped for opencast dewatering in the Rhine District, but the quantity of water to be lifted is increasing steadily and is expected to reach 1 400 000 000 m³/yr, the highest figure, by 1966. While the waterlifting to coal output ratio was still 2:1 in 1955, it will be more than 15:1 in 1966. A special channel 20 km long has been constructed to take this underground water into the Rhine, the present rate being 25 m³/sec. The existing drainage channels have been improved.

SURVEYING

The concentration of brown coal output on a few opencast operations necessitates mining on a large scale within them. As open-cuts become deeper, so surveying is becoming more complicated. Mining companies have therefore given up the old methods of surveying usual for opencasts and have introduced aerial photographic methods. In conjunction with electronic evaluation this method makes it possible to measure all changes so exactly that hereafter even the overburden moved and coal produced can be calculated. The principal advantage of this method compared with conventional surveying is the considerable saving of personnel and time.

CHANGES IN LANDSCAPE

As a consequence of opencast brown coal mining in the Rhine District more than 130 km² of the surface have up to now been taken over, of which 60 km² have been restored. Of these, 34 km² have been reafforested (Fig. 9), 16 km² have become agricultural and 10 km² have

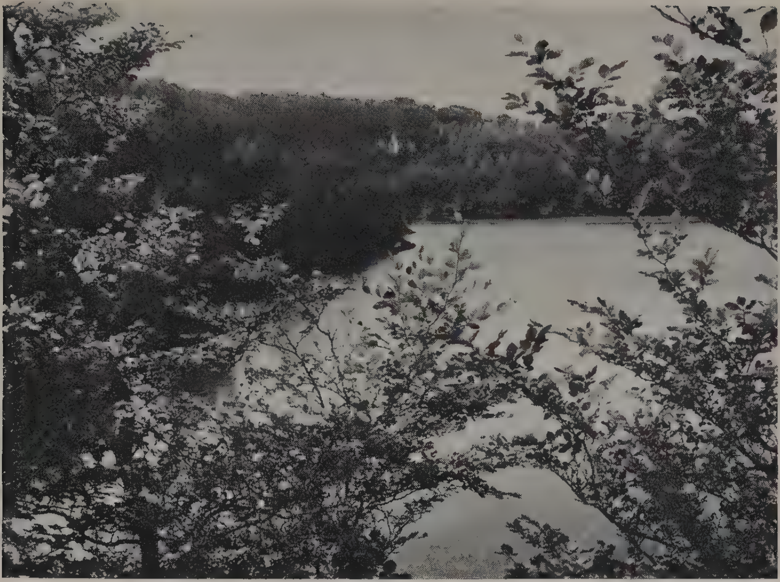


Fig. 9.—Afforested worked-out area with lake.

found other uses. More than 70 km^2 are necessary for the mine area, and villages, roads, railways, etc., in this region must be moved in time because extension of the opencast cannot be limited by surface conditions, only by geological conditions. Nine villages have so far been moved in the Rhine District and twenty others will have to follow within the next ten or twenty years, involving altogether about 25 000 people. Such small villages gain technically since they are rebuilt to modern standards. To date, more than 50 km of road have been moved and some 100 km will follow. Certain railway lines have been replaced while others have been closed down owing to rationalization in railway transport.

In order to leave no devastated land behind, the mining companies are bound by regulations to restore the landscape and try to regain the former standard of cultivation as far as is possible (Fig. 10). In areas which were wooded before mining started new forests are to be grown, and those of mainly agricultural use are to be similarly restored. The large quantities of loess loam in the opencast regions make this possible, and the dumps which are to be recultivated for agricultural use can be covered with a layer of this loam 1–2 m thick. After intermediate sowing of soil-improving plants, and systematic use of artificial as well as natural manure, outputs are possible which are not less than that before

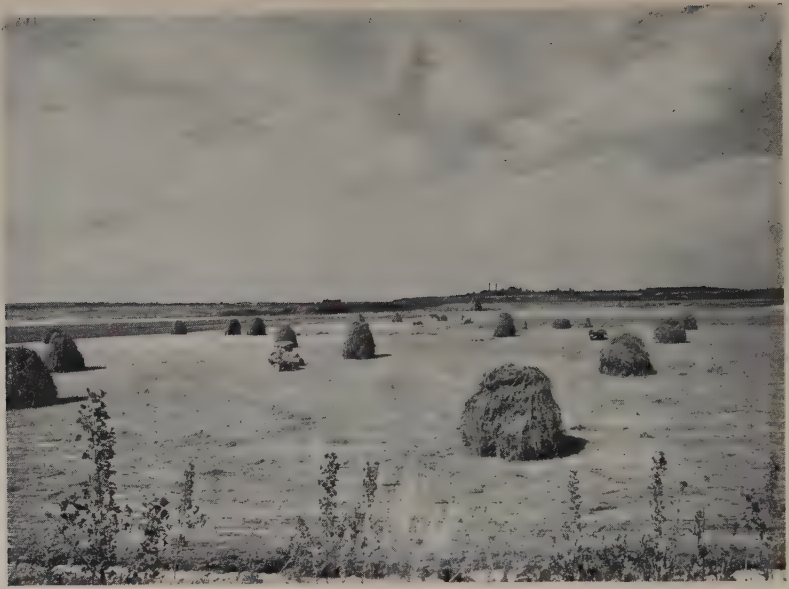


Fig. 10.—Landscape after restoration to agriculture.

mining started, even within three years of beginning cultivation. These areas are then taken over by the farmers who had to leave their land while mining was in progress.

Owing to the size of modern opencasts the areas involved for agricultural recultivation are so big that they allow reallocation in accordance with the latest scientific and economic principles. Windbreak hedges cross the agricultural area by plan, the dumps are afforested, and, in order to regain woods and build up a green zone with lakes in between for the town and industrial population, great districts have been planted mainly with poplars mixed with alders, larches, acacias, red oaks, and maples (Figs. 9 and 10).

COSTS

Owing to the high standard of technical equipment the cost structure of the brown coal industry is vitally affected by capital costs, while labour costs, which at present are about 30 per cent of the total, are losing importance. Using modern equipment, costs can be held to such limits that raw brown coal as a primary energy source can maintain its privileged position. The advantageous terms for producing primary energy from raw brown coal is the basis on which black coal is allowed to participate on a larger scale in energy production so that the resulting combined costs for brown coal, black coal and water power still lead to a favourable price for electric power.

The investment costs for opencasts approach 25–35 DM/t production per year depending on extent, thickness of overburden, and formation of coal deposits, and the investments in the Rhine District, giving the stated output, have exceeded a total of 2 000 000 000 DM during the last decade.

Initial costs for opencast machinery are calculated on their service weight: 6 DM/kg weight of excavators for the big bucket-wheel dredges including the electro-technical equipment and about 1 DM lower for spreaders. The initial costs of a conveyor system depend on the width of the belts, and include the cost of the structural parts together with the driving stations as well as the belts, which constitute up to one-third of the total. They amount to about 4500 DM/metre conveyor length for a belt 2·20 m wide, and to about 1300 DM/m for a belt 800 mm wide, i.e. the wide belts are about 20 per cent more expensive per m^2 conveyor belt surface than the narrow ones. One railway unit, consisting of 1 locomotive and 10 overburden wagons with a capacity of 96 m^3 each, costs about 2 200 000 DM; 1 km double track with ‘S 64’ rails, including the necessary works, costs up to 2 000 000 DM and a dewatering well costs about 200 000 DM, including equipment.

Besides the high investment costs there are further expenses for purchasing the necessary agricultural areas and forests and their recultivation. The cost of removal of roads varies between 400 000 and 4 000 000 DM/km, depending on the condition and position of the road, while removing villages costs between 50 000 and 100 000 DM per household.

Operating costs vary considerably between opencasts and depend on the geological and technical conditions. The costs for overburden per ton of coal are important and these depend mainly on the ratio of overburden to coal within the opencasts. Winning and dumping account for 25 per cent each, transportation 40 per cent, while the difference of 10 per cent is for other running costs.

Coal winning and transport to the consumers have also to be considered in calculating operating costs, in addition to dewatering, and extensive work in connection with the removal of villages, roads, and railways before laying claim on the ground, as well as the cost of restoring and recultivating exhausted open-cuts. It is the aim of technical progress to keep these costs down, in spite of the worsening of the opencast conditions, by virtue of technical development and further rationalizing measures. These comparatively low costs are the basis of the fuel supply costs of the brown coal power plants and reflect in the economically favourable electric energy produced here.

Lighting of Opencast Coal Mines

with an appendix on the lighting of railway sidings

W. B. BELL, B.Sc., Assoc.I.Min.E., M.A.P.L.E.
University Lecturer

P. E. O'SULLIVAN, B.Sc., Ph.D., Assoc.I.Min.E.
D.S.I.R. Research Fellow

Department of Mining Engineering, University of Newcastle upon Tyne

---

SYNOPSIS

The paper reviews the current practice in lighting opencast coal mines. The results of surveys are depicted by a general example incorporating large individual machines as well as conveyors. The number and types of lighting fittings are given, together with the resultant installation illumination values. Finally, an assessment with respect to lighting and safety and a lighting recommendation are made.

The paper concludes with an appendix on the lighting of railway sidings.

The paper contains the results of a series of surveys on current lighting practice in opencast coal mines throughout Britain, the initial purpose of which work was to provide information for the Commission Internationale d'Eclairage on the safety aspects pertaining to opencast coal mining at night. The original enquiry came from Poland, the Mine Lighting Secretariat at that time, who were interested in the fact that no reportable accidents had occurred at night in British opencast coal mines, whereas such accidents were frequent abroad.

GENERAL SITUATION

With the exception of the dual-purpose equipment mentioned later, there have been no new developments in the lighting of opencast mining sites since 1955. The lighting of sites follows a somewhat general pattern and a typical installation is described.

Experience has shown that since all the machines are constantly on the move, and all workers are either in vehicles or on machines during the hours of darkness, the most effective illumination is obtained by mounting the lighting fittings on the machines. In addition to providing the necessary illumination for the various tasks which have to be performed, the areas immediately around the machines are adequately lighted.

INSTALLATION

The largest machine, the walking dragline, executes the main cut, and is approximately 70 ft (21·34 m) wide by 90 ft (27·43 m) long, with a boom of approximately 300 ft (91·44 m) rising to 100 ft (30·48 m) above ground level.

Lighting fittings arranged in groups of four are fixed on the boom and focused on the immediate area surrounding the dragline of the bucket. The fittings are of the direct concentrating type fitted with reflectors of anodized aluminium and housing 1000-W tungsten-filament light sources (Fig. 1). A further similar type of fitting is fixed to the top of the boom and focused along the line of the rope to the bucket.

Small variations exist from site to site since the length of the boom is determined by the size of the bucket. On a boom 300 ft long (91·44 m), nine fittings are provided and these give the following illumination at

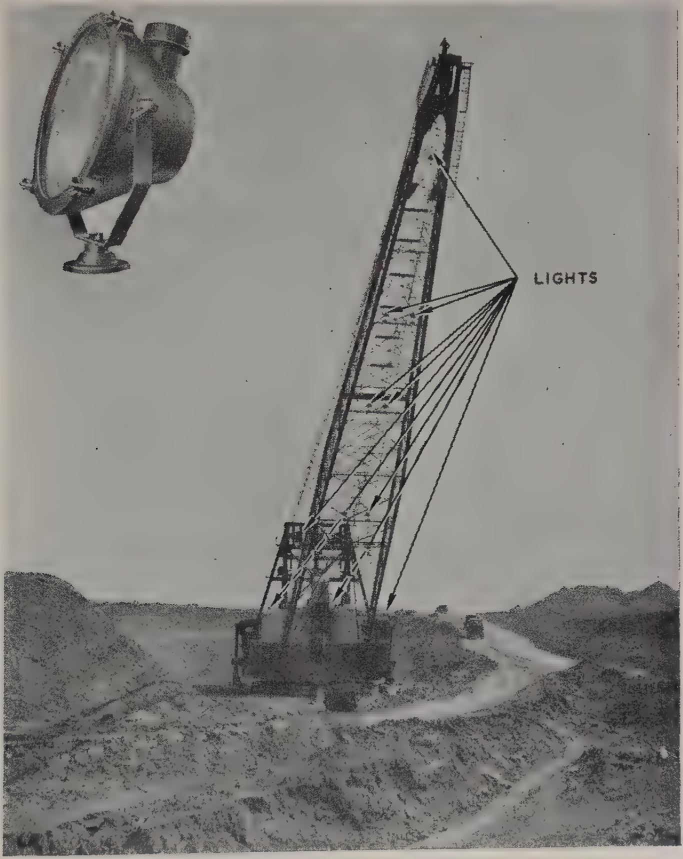


Fig. 1

ground level: 6·5 lumens/ft^2 at the point where the underside of the boom is nearest to the ground, varying down to 1·3 lumens/ft^2 at the point where the boom is farthest from the ground.

Two aircraft warning lights are provided on the top of the boom.

The immediate area around the dragline house is illuminated by four roof-mounted dispersive-type fittings having polished reflectors (Fig. 2)

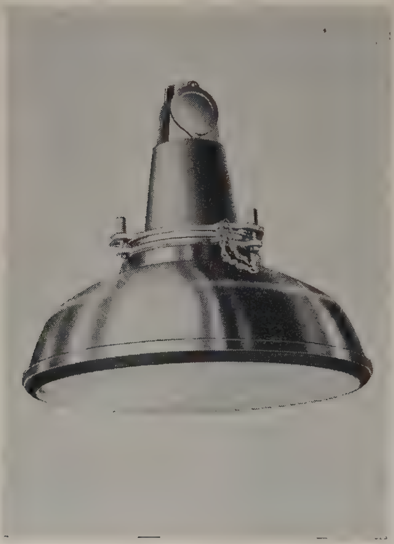
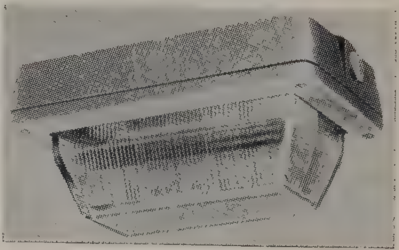


Fig. 2

Fig. 3



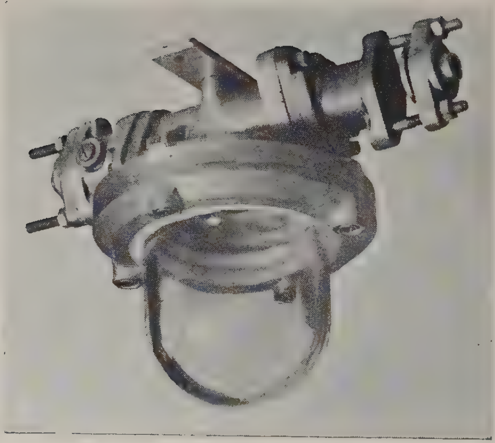


Fig. 4

and four similar fittings are mounted on brackets, one in the middle of each side wall. All eight fittings incorporate 500-W tungsten-filament light sources.

The average illumination achieved at ground level in the vicinity of the dragline house is 4·5 lumens/ft^2.

For maintenance purposes four bulkhead-type fittings incorporating 100-W tungsten-filament light sources (Fig. 3) are fitted underneath the dragline house and the illumination achieved at ground level is approximately 3 lumens/ft^2.

The interior of the dragline house is illuminated by six dispersive-type fittings with polished reflectors and incorporating 500-W tungsten-filament light sources, together with four parabolic angle-type fittings with polished reflectors and incorporating 150-W tungsten sources. Two further fittings of the pendant type and incorporating 100-W tungsten sources (Fig. 4) illuminate the stairway to the operating cab and the cab itself. The average illumination achieved at floor level in the dragline house is 4·5 lumens/ft^2.

A bucket-wheel excavator is used for the removal of overburden and the main lighting on the machine (Fig. 5) is provided by four floodlight fittings incorporating 500-W tungsten sources. In addition, six bulkhead-type fittings incorporating 100-W tungsten sources are arranged

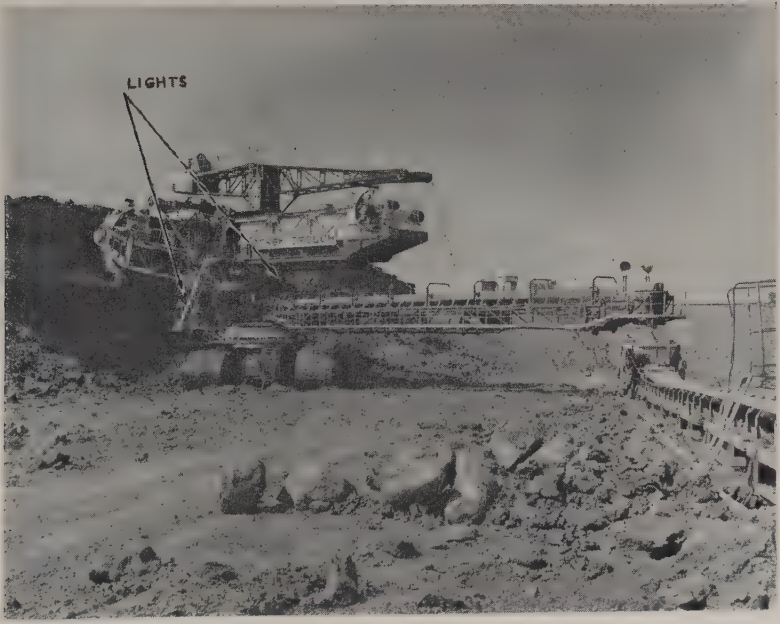


Fig. 5

at intervals on the catwalks. The illumination on the task varies from 4 lumens/ft^2 at the top of the working face to 0·15 lumens/ft^2 at the bottom due to shadows cast by the bucket wheel.

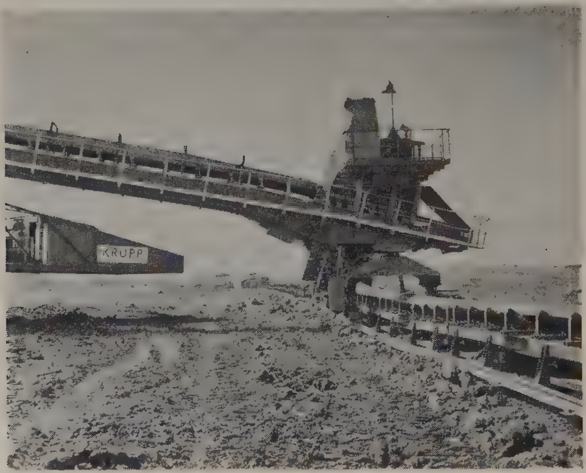
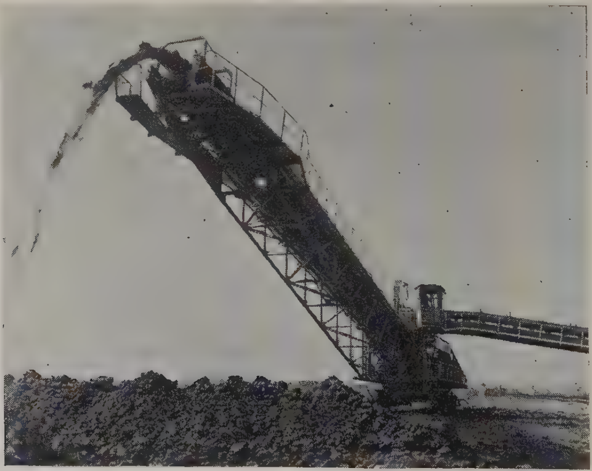


Fig. 6(a)

Fig. 6(b)



When a main conveyor is arranged around the site, illumination is only provided at transfer points and at the dirt disposal end (Fig. 6). Each of these locations is illuminated by a floodlight fitting incorporating a 500-W tungsten source. Consideration is, however, being given to providing illumination along the whole belt in order that obstructions causing stoppages of the conveyor may more easily be seen.

Bulldozers and lorries are fitted with conventional 36-W headlights and automatically operated reversing lights. A typical coal lorry is shown in Fig. 7.

Fig. 7

Dual-purpose lighting and welding units, equipped with headlamps for inspection purposes, are now in the process of being manufactured (Fig. 8). Each unit comprises a welding generator and a heavy-duty low-voltage lighting generator driven by a diesel engine, the whole unit being mounted on a four-wheel chassis. Floodlights are mounted on hinged poles (to be lowered when travelling) and beam from 15 ft where erected. Each floodlight houses a 500-W reflector-type lamp and it is assumed that the illumination to be provided will be sufficient for repair operations to be effected.

Handlamps incorporating 60-W tungsten sources are energized at 24 V and six-socket outlets are available. A mechanical control from the gearbox permits selection to be made for welding in the daytime and lighting at night.

In the installations visited, none of the coal-getting operations, including drilling and blasting, was performed at night; therefore no special lighting was necessary.

COST

In general, as the lighting fittings were mounted on the machines themselves, the cost of both the installation and the maintenance are infinitesimal compared with the machine. Again, as the power consumption was negligible compared with that of the machine no breakdown of costs has been attempted.

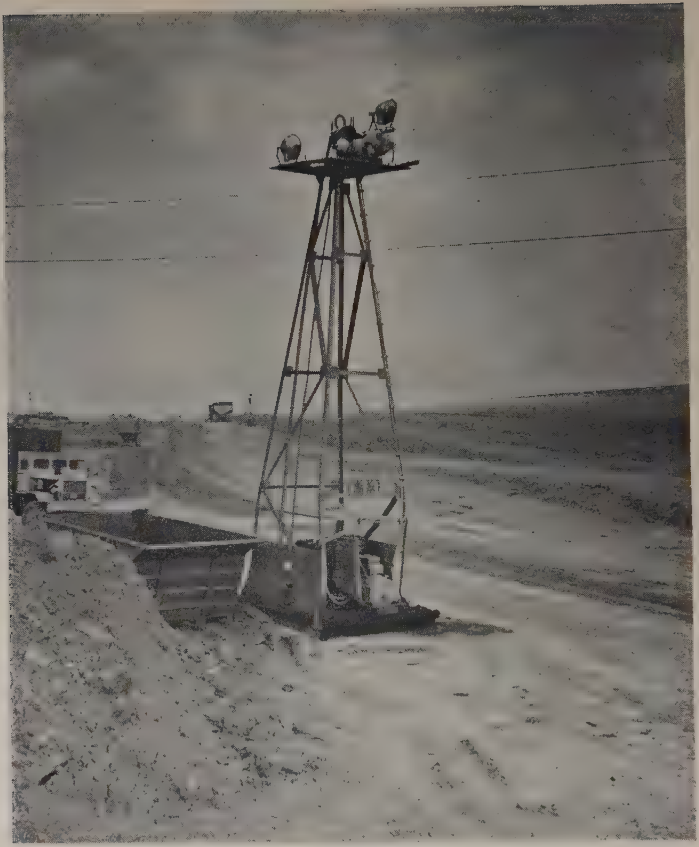


Fig. 8

CONCLUSIONS

Although the measured illumination values are relatively low compared with normal industrial practice, the lighting fittings are so placed that the maximum benefits can be obtained from them, giving fairly good seeing conditions.

Because all the light is in the vicinity of the machines, workmen tend not to stray from the machine area and so fall down the 'cut'. Again, the machines themselves provide a measure of safety against landslides and 'slips'.

Where a.c. power is available, the more efficient long-life medium- and high-pressure colour-corrected mercury vapour lamps should be used; these will increase the illumination values by 2 to $2\frac{1}{2}$ times for the same wattage consumed.

Appendix

THE LIGHTING OF RAILWAY SIDINGS

Until a few years ago it was the practice to use fittings incorporating tungsten-filament light sources mounted on short columns for illuminating railway sidings. The first departure from this method was the introduction of the more efficient sodium light sources. Later, floodlight fittings mounted on 50-ft (15·24-m) columns were introduced to light smaller sidings of the long and narrow type. Also introduced at the same period were street lanterns mounted on 25-ft (7·62-m) columns. Both methods have proved successful in service and are still used. Such installations are also necessary where tracks run in cuttings, or between slag heaps, and where obstructions make lighting from higher towers ineffective.

It has been shown that there is an economic advantage by lighting large railway sidings by floodlights mounted on towers of up to 150 ft (45·72 m) in height, and the use of this method is increasing. Several installations are now in service giving better lighting, especially between tracks, than by any other method. Installations use 100-ft (30·48-m) or 150-ft (45·72-m) towers sited near the weighbridges and light flux is directed to either side along the directions where the rails fan out. Between the towers the light flux is from two directions and this provides better visibility. The end sections are lit from one direction only, it being considered that extra towers to provide two-way lighting would be too expensive. Each tower normally carries between 25 and 35 projector-type floodlight fittings incorporating 1500-W tungsten-filament light sources. The upper fittings provide the narrow-angle beams for the longer throws, the middle fittings the medium, and the lower fittings the wide-angle beams. Spreader glasses are added to light the areas around the base of the towers. Where convenient and necessary, extra floodlight fittings incorporating 1000-W tungsten-filament light sources may be mounted on any buildings adjacent to the sidings at a height of about 30 ft (9·14 m) in such a manner that the light flux from them is directed towards the towers to counteract shadows cast by these buildings.

The illumination at ground level at points and crossovers is between 1 and 2 lumens/ft^2 and over the general track area the average value is 0·3 lumen/ft^2.

At one particular siding an installation at 50-ft (15·24-m) mounting height of trough-type floodlights, each containing two 400-W mercury lamps burning horizontally, is giving satisfactory service. The towers are at a spacing : height ratio of 4·5:1 and carry 2, 3, or 4 floodlights.

They are sited to give two-way directional lighting and the average illumination at ground level is 0·5 to 1·0 lumen/ft².

Street-lighting lanterns used for low-mounted installations are now of the medium-angle beam or cutoff type and incorporate either 140-W sodium-light sources or 400-W mercury-discharge lamps. Spacing is between 95 ft (28·96 m) and 120 ft (36·58 m).

Columns are usually sited at points and crossovers and the illumination at ground level near to the columns is 3·0 lumens/ft². The average illumination for the area is between 0·5 and 0·9 lumen/ft².

Future Trend in High Tower Lighting for Sidings

For the larger marshalling yards and sidings the use of high tower lighting is expected to increase. At present tungsten-filament light sources are in general use, but the future trend is likely to be towards the application of the high-wattage mercury-discharge lamps, probably of the 1-kW type. The advantages of this particular lamp are (*a*) the high light output and (*b*) the very long life of 5000 hours.

One 1-kW mercury-bulb fitting gives approximately the same light output as two 1500-W tungsten fittings, hence the number of fittings and the total loading can be reduced with consequent economies in cabling, switchgear, tower, running and maintenance costs. The mercury lamp is more expensive and requires control gear, and since the cost per fitting is approximately equal to that for two fittings using tungsten sources, the overall fittings cost is the same.

For area lighting it is usual to burn the mercury lamp horizontally because of the long and narrow source shape. This produces a narrow beam in the vertical of about the same beam angle as that provided by a floodlight with a tungsten source, and a wide beam horizontally of about three times this width. The light distribution from the mercury lamp is therefore best suited for lighting the larger marshalling yards and areas which are fairly wide.

Acknowledgements.—The authors wish to acknowledge the assistance of the following: The National Coal Board Opencast Executive, North Eastern Region; The C.I.E. Mine Lighting Sub-Committee of Great Britain; and Mr. R. Buffrey, Lighting Engineer to the National Coal Board.

Discussion

(Papers 28–30)

Mr. D. G. Hemmant,* introducing Papers 28 to 30, said he would like to deal first with Paper 29, 'Opencast technique in German brown coal mining' by Dr. W. Tilmann, because it was the most general and comprehensive of the three—the other two being descriptions of individual operations and installations.

Brown coals of Germany, especially those of the Lower Rhine district, and indeed many of the lignites of the world, were characterized by comparatively low calorific value, usually mainly due to high moisture content, comparative physical softness and the loose unconsolidated nature of most of the overlying strata. The deposits of the Lower Rhine area were also fairly massive and when mining started were comparatively shallow and easy of access. 'Shallow' was perhaps an ambiguous word, but the ratio of overburden to coal in the early mines varied up to 0·5 to 1. Although brown coal itself was often relatively impermeable, the overburden was not, and water presented a considerable potential hazard with increasing pit depth, as well as a major economic problem. The low calorific value of the brown coal and the small productivity of underground mining inhibited such a method on economic grounds, except in some special cases. But fortunately the relative softness of the brown coal itself, and still more that of the unconsolidated overlying strata, assisted mining by opencast methods. The low calorific value meant that the market for the raw material was essentially local and required a low cost of production. That required large-scale operations and much detailed planning—probably necessary in any event, but certainly at any considerable depth and when water had to be pumped in large quantities.

Because of its low calorific value brown coal had to be processed locally, either to electric power, briquettes or chemicals, which could be transported economically over greater distances, although their availability might well have attracted industry to the neighbourhood and reduced the average distances over which such transport was necessary.

In the Lower Rhine area suitable market conditions were well established, but it was still necessary to reduce the cost of production to the minimum faced with cost-increasing tendencies such as increasing depth, increase in the make of water and higher costs of labour and materials. That had been done by developing specially large machines, which were getting still larger, for both mining and transport.

---

*Truscon, Ltd., London.

The comparatively soft nature of both the brown coal and the overburden had favoured the use of continuously operating excavators. The earlier machines were bucket-chain dredges which either, like floating dredges, operated from above on material below the working level, or as slewing dredge enabling a high as well as a deep cut to be made. Over the last fifteen years or so those had tended to be replaced by bucket-wheel excavators which were more flexible, more selective and more efficient—the function of the buckets being mainly to cut and excavate, and not to convey, the latter function being performed by conveyors designed for the purpose. Their continuous action called for continuous transport, which had, of course, to match the capacity of the excavators themselves—up to 8700 m^3/h. Some conveyor systems were even capable of nearly twice that figure—15 000 m^3/h. Those consisted of steel cord conveyor belts 2·2 m wide, travelling up to 6·5 m/sec (in English measurements 7·3 ft and 1300 ft/min). Most of those machines were tailor-made for particular projects, their maintenance and movement also calling for the use of specialist machines and systems of communication.

He did not want to give the impression that all the emphasis was on the use of conveyors for transport. In fact, the paper pointed out that 82 per cent of the transport was by rail, with only 17 per cent by belt conveyors and 1 per cent by other means. The rail transport was also on a very big scale, wagons up to 90-ton capacity being used for coal and 180-ton capacity for overburden. In both cases axle loads were up to 30 tons, requiring the use of rails with a weight of 64 kg/m, equivalent to about 128 lb/yd.

Traditional methods of surveying were inadequate for such an enormous operation and aerial photography was used. Land reclamation was also dealt with in the paper and when whole villages had to be moved the inhabitants often derived considerable benefit from the improved houses which replaced the old ones.

The result of all the highly mechanized working was that an overburden to coal ratio of up to 6 to 1 could be achieved economically with a productivity of more than 50 tons per man-shift.

Reference to bucket-wheel excavators with a crowding action was made; he understood, however, that that was not a true crowding action but a means of retracting the bucket wheel after completion of one horizontal slice in order to start another, the actual feeding of the wheel into the work being by slewing, resulting in a milling action.

He would not be doing justice to the paper if he did not refer to the water problem, although that type of problem had been dealt with in much greater detail in Paper 18 (pp. 390–415). In the Lower Rhine area 1 400 000 000 m^3/yr were expected to be pumped from 550 wells by 1966, and the ratio of water to coal would have increased to more than 15 to 1, compared with 2 to 1 in 1955. The effect of the drawdown was felt at a distance of 20 km.

The specialist knowledge gained had been of great value to other countries with similar problems, and in the last 15 years out of more than 200 bucket-wheel excavators constructed nearly half had gone to Europe, India, Australia, Indonesia, Africa and Central America. It was reasonable to assume that the effect of the development work in high-speed conveyors had travelled even more extensively. No doubt additional experience had been gained in those overseas installations owing to the different nature of some of the conditions.

'Opencast coal mining in Great Britain: operation of Acorn Bank and Westfield sites', by Mr. W. G. Martin, described two installations in the United Kingdom—one a strip mining site in which three or more seams were worked, the other more like a quarrying operation, in which the advantages of scale had been similarly appreciated. However, Great Britain did not offer the same scope for large-scale opencast coal mining as Germany, and for various reasons, some connected with the need for blending to produce a specific grade, less concentrated operations were common in opencast mining for other minerals.

The much thinner, although more valuable seams, the harder overlying rocks, the fact that many of the shallower areas were worked, albeit sporadically, in the days before large-scale excavators were developed, and the comparatively low cost of deep-mined coal until the last war, resulted in areas suitable for large-scale opencast working being relatively few and limited in extent. Nevertheless, in the two installations described the deposits were large enough to justify the installation of equipment not normally used in the United Kingdom—an original system of derricks for hoisting the coal from the bottom of the cut to the surface and a system of high-speed conveyors and slewable spreaders for handling the overburden, similar in principle to those referred to in Paper 29.

For both operations ammonium nitrate–fuel oil was used for blasting initially, although slurry was later introduced at Westfield owing to difficulties with wet conditions. The slurry was mixed at a factory on the site. The high-speed conveyors at Westfield conveyed the overburden, after it had been crushed, at the rate of 3440 metric tons per hour—a far bigger capacity than any conveyor used in opencast mining in the United Kingdom before.

Reference to the problem of restoration was made, and also to the great importance of satisfactory site communications, the relevance of which to maintenance and to the reduction of dislocations caused by plant failures was obvious.

A further point of interest was the fact that a special coal-haul road was constructed at Acorn Bank in preference to using smaller haulers on public roads.

The paper also contained some very interesting comparisons between the costs of tyres, drilling bits, and bulldozer tracks at the two sites and showed that costs were appreciably higher, in varying degrees, at Westfield than at Acorn Bank. Explanations for the increases were given.

Paper 30, 'Lighting of opencast coal mines (with an appendix on the lighting of railway sidings)' by Mr. W. B. Bell and Dr. P. E. O'Sullivan, was clearly relevant to Papers 28 and 29 because the installation of such expensive machinery required that it should be used as many hours per day as possible. Furthermore, with such highly organized operations, safety must be pre-planned and not merely left to common sense, however useful the latter undoubtedly was. An installation involving a large dragline and a bucket-wheel excavator was described, as were dual-purpose lighting and welding units now in process of manufacture.

Mr. T. T. Heywood,* referring to Paper 30, said that Fig. 8 (p. 664) showed a lighting tower. When working with lighting towers some years ago it was found that the lamps suffered damage, particularly filament failures, owing to the vibration and jarring which occurred when towing the towers from position to position. Those failures were much more frequent than lamps which burned out after a normal life. He wondered whether lamps had been developed which gave greater mechanical strength to the filaments to help them stand up to rough handling.

Mr. D. E. Speight† said that Mr. Bell and Dr. O'Sullivan referred exclusively to tungsten-filament lights. Their comments on the use of mercury-vapour lamps, with their higher output per kilowatt, and also on colour correction, would be welcomed.

Mr. A. Major-Stevenson‡ asked if the authors of Paper 30 had had any experience of using 200-W lights for low voltages, say 24 V. Any performance data would be of great interest.

Mr. S. W. L. Pardoe,§ speaking on Paper 28, said that in his opinion the most interesting points concerning the Acorn Bank site were (1) that it was started by a 10 000 000-yd^3 box-cut at the deep side, of which 4 000 000 yd^3 could be moved across a road to permanent spoil on another site then under restoration, and (2) the method of raising coal from the cut by derricks. Both those factors arose from the desire for progressive restoration of the surface.

---

*N. V. Mijn Maatschappij, Curaçao.

†Bowmaker (Plant), Ltd., Staffordshire.

‡Ruston-Bucyrus, Ltd., Lincoln.

§National Coal Board Opencast Executive.

Three 120 RB shovels and two 1150B draglines were used to excavate the box-cut—all loading into 22-t (Type B1.TD) Euclid wagons, the largest available in 1955. The draglines might not have been chosen for the work, but were going to be needed later for the main excavation and their use saved the wagons having to climb out of the deep cut. He understood that the 22-yd^3 bucket was used to load directly into the Euclids, while the 25-yd^3 bucket discharged on·to a strong sloping steel plate which both protected the wagon's cab and broke the fall of the material being loaded. In spite of that rough treatment the contractor's fleet of Euclid wagons gave more than 18 000 working hours before replacement with new larger types. From time to time during the working of the site, and particularly when going deeper for the Top Yard and Yard seams, the 1150B draglines had been used to load the 27-t Euclids directly, without any intermediate chute. Under those conditions the cycle time could be less than 60 sec and the output some 850 yd^3 per working hour. A D8 Caterpillar bulldozer was kept busy clearing the considerable spillage. In the box-cut, using the smaller wagons, the output was down to about 500 yd^3/h.

Much that the National Coal Board did on its opencast sites was concerned with making them acceptable to the public in a heavily populated country. Acorn Bank, like most of their sites, was to be restored to full agricultural use with no long-term loss of fertility.

It was important to occupy as little land as possible. To minimize double-handling of overburden with deep excavations of relatively small surface area they had to work with the smallest volume of cut that would allow the efficient use of the machinery needed to give the desired output. That meant that they were anxious to avoid access ramps of large volume which must add to either the initial or the final void—material that would have to be moved to or from a temporary dump. He doubted if similar conditions prevailed in other countries and would be interested to hear if anyone had considered tunnels advancing through the spoil banks or derricks to lift coal nearly to surface level. At present, the problem of how best to raise coal from deeper excavations than Acorn Bank (more than 300 feet below a fairly level surface) was being considered and any other ideas would be of great interest.

The derricks were successful, but they must have caused the contractors considerable anxiety. The obvious need for a stable high wall was mentioned by Mr. Martin, as was the method of turning the bogies on to cross-tracks to move to successive cuts. It was that operation, in particular, that he would want to examine if he were considering the use of derricks elsewhere, for it was clear to an onlooker that it took a great deal of time and labour.

He had no personal connection with Westfield and would only say of that site that he knew of nowhere else where overburden was being crushed and conveyed. At the Radar North site a conveyor and spreader system was being used with a bucket-wheel excavator to move only drift clay, relatively free of boulders. He believed that the transfer points there had caused as much trouble as any other part of the system, the clay so easily clogging the chutes. He supposed that the crushed rock at Westfield would flow more freely, but also more abrasively, and was sure that they would be interested to hear in more detail how the conveyor system, and particularly its transfer points, had appeared to the contractor who had been operating it. Incidentally, at the Radar site the coal was hauled from the bottom of the cut, approximately the same 180 ft average depth as Acorn Bank, by 14-t four-wheel-drive wagons.

Mr. K. S. W. Orr* said that Mr. Martin stated that Caterpillar D8 tractors were used at both Acorn Bank and Westfield and that track costs were 35 per cent higher at Westfield (see pp. 635–6). Even allowing for more abrasive conditions, that percentage appeared very high. He wondered if the author was comparing similar model D8s equipped in a similar manner, i.e. was he comparing two 22A-series machines equipped with DS blades. Even comparing identical units, a realistic method would be on an undercarriage cost/yd^3 of material moved basis.

At Westfield, Caterpillar D8 tractors equipped with dozers alone and D8s equipped with dozers and rippers were operated. Information on the comparative track costs of those arrangements would prove very interesting. Further, were the ripper-equipped D8s used to rip and doze rock overburden? If so, he would welcome details of costs of removing rock overburden by that method, compared to drilling and blasting. In the U.S.A. crawler tractors with rippers were much more widely used than in the United Kingdom for removing rock overburden. When the rock bed was too much for the ripper alone one of the techniques employed was pre-blasting followed by ripping. That method apparently resulted in a lower cost/yd than by pure blasting. He would be interested to hear of any experience of that technique.

In the type of work described, what did the author consider an economic replacement cycle for his dozers?

---

*Caterpillar Tractor Co., Ltd., Scotland.

Mr. B. J. Greenland,* dealing with Paper 28, said that the section (p. 627) on blasting brought out very well the advantages of slurries versus AN–FO mixes where large quantities of water were met. To establish a better idea of the competitiveness, from the price point of view, of slurry, could Mr. Martin give some idea of ton/lb with AN–FO and ton/lb obtained with slurries. As those figures were really only half of any blasting story, the other half being the drilling costs, could he also give some idea of overall drilling and blasting costs for AN–FO and slurry.

He would certainly agree with the author's comment that the 'regulations governing the manufacture [of a slurry explosive using a nitroglycerine base] on site are stringent', and that only on a long-term contract could the cost of setting up a suitable factory be contemplated. Perhaps the author would give his opinion as to the minimum amount of explosive that would make such an enterprise an economically viable unit.

The problem of on-site mixing was complicated by the fact that the slurry used was a nitroglycerine slurry and the procedure adopted was, in fact, one where the nitroglycerine was provided, suitably stabilized, and the other ingredients were added on site under strict control and according to the Explosives Act.

Mr. E. S. Chaplin,† referring to Paper 28, said that there were in all, at Westfield, 22 different and separately recoverable coal seams or leaves of coal seams. Over the greater part of the area currently under development the aggregate thickness of the *in situ* coal was approximately 128 ft, but after deducting shale and fire-clay partings some 82 ft in thickness of recoverable raw coal remained.

The author mentioned that the coal seams were lying at an angle of anything up to 45° from the horizontal and that the method of mining was fundamentally a successive lowering of the pavement of the pit by benching to an average height of 15 ft. That method of working was very effective, but because of the steep angle at which the coals were lying, and because of the many faults, it did mean that in achieving a very tight blending specification there were usually several occasions during an average shift when an excavator temporarily went out of production while being re-located or moved to a different face before commencing recovery of a coal of another quality. Of the eight principal excavators at the mine (four 150 RB 5-yd^3 shovels and four 54 RB $2\frac{1}{2}$-yd^3 shovels) a fair assessment of the daily pattern was that five machines would be digging coal or partings and three machines would be digging overburden. To serve those eight 'prime movers' a fleet of some 30 dump trucks was in operation. With very few exceptions the trucks were equally suitable for carrying raw coal to the reception hoppers at the disposal centre or shale and rock to the crushers for loading to the overburden conveyor system.

---

*Explosives and Chemical Products, Ltd., Derbyshire.

†National Coal Board Opencast Executive.

Once the shift had begun any incidental interruption of the digging cycle of any one of the eight loading excavators—whether it was because a machine completed one face task and had to travel to a different face, or because a hoist-rope broke—required an immediate reassessment to determine which of the seven other 'prime movers' could best temporarily utilize the services of the 3 or 4 dump trucks made idle owing to their parent loader going temporarily 'on stop'. Similarly, after the start of shift one dump truck might become a casualty and have to leave the field, thus leaving its parent loading excavator starved of transport. An assessment of whether that was the most acceptable position 'job-wise' or whether one of the other seven excavators should yield up one of its dump trucks in favour of the first excavator had then to be made.

A well placed observation office at Westfield gave a panoramic view of the site, but even in full daylight a central observer could not see *all* the working faces and in the early morning or late afternoon in winter the outlook was limited that much more. In the pit itself it was very difficult for an observer to see more than, perhaps, two or three of the excavators from the same vantage point and it could well take an experienced foreman some 15 to 20 minutes to visit all the faces to assess which one could best utilize an additional dump truck or, alternatively, if a 'prime mover' had broken down, which of seven faces could best share an unexpected addition of three or possibly four dump trucks. He understood that a dump truck cost some £18 000; if there were thirty on the job the transport allocation between excavators need only fall $3\frac{1}{3}$ per cent below 100 per cent efficiency and, in effect, an unnecessary 'passenger' truck was being carried.

Perhaps the author would care to say a few words on the methods and practice currently employed in determining the short-term allocation of dump trucks in the pit at any given period during a shift, and whether he considered that aspect of the work lent itself to control by a simple electronic computer designed to fit the characteristics of the coal and overburden transport logistics at Westfield.

Mr. M. W. Rushton,* commenting on Papers 28 and 29, said that power-shovel rubber-tyred vehicle methods were described, using machines of various capacities. He wondered if the authors could give some formula, empirical or otherwise, for matching the size of vehicle to the size of the power-shovel digger or the digging rate of the shovel.

---

*Nchanga Consolidated Copper Mines, Ltd., Chingola, Zambia.

The matter was highlighted by the fact that the capital cost of shovels was written off over a period of years, in some instances 20 years, whereas the life of rubber-tyred vehicles was reckoned in terms of thousands of hours, usually 20 000 or so.

WRITTEN CONTRIBUTIONS

Mr. G. J. Shrimpton:* I would like to ask Mr. Martin by what amount the hole spacing is increased and to what extent do the reduced drilling costs offset the increased explosive cost when using the TNT and nitroglycerine-based slurry explosive.

Mr. R. O. Morris:† I wonder if Dr. Tilmann has considered the possibility of freezing groundwater along the margins of opencast workings. That would mean that pumping would only be required to drain the actual working areas, it would eliminate the effects on groundwater over large surrounding areas and would produce firm and stable margins to working areas which would be safer and might result in narrower working areas.

Mr. N. R. Cameron:‡ I recently visited the brown coal field described in Paper 29 and, besides being struck by the enormous scale of operations, as a geologist, was agreeably surprised at the amount of wood, often in the form of complete trees, within the lignites.

In view of the number of trees seen, admittedly at only one level in one mine, I wonder what effect their presence has on the efficiency of operation of the chain bucket excavator referred to by Dr. Tilmann.

I imagine the trees tend to be concentrated at certain horizons and that working methods are adopted according to their frequency of occurrence.

---

*Camborne School of Mines.

†L. M. Van Moppes (Diamond Tools), Ltd.

‡Ministry of Housing and Local Government (Geology Section), London.

Authors' Replies

(Papers 28–30)

Mr. W. G. Martin: With reference to Mr. S. W. L. Pardoe's contribution (pp. 670–2), the conveyor system in use at Westfield has certainly presented a number of problems in operation. Since the roll crushers can pass rock as large as 24 in. by 30 in. by 10 in. and the conveyors operate at 800 ft/min, the transfer points and the conveyor belts are subjected to very heavy wear. The problem is further complicated by the fact that it is often necessary to convey soft wet partings over the system. Unfortunately, a transfer point which is designed to handle large pieces of hard rock and place them as gently as possible on the conveyor belt will clog and fill to overflowing in three or four seconds when soft wet partings are put through the system. The design of the transfer points has been modified to produce the best compromise, but since it is wholly unacceptable that a transfer point should ever overflow, it is necessary to accept as inevitable the fact that the rock will be thrown on to the belt sufficiently hard to cause damage.

The impact plates at the transfer points have been modified to reduce the frequency of renewal. Originally, these plates were faced with close-spaced 25-lb rail. These have been replaced with an egg-crate construction made up of $\frac{1}{2}$-in thick bars in 6-in centres, the bars being 4 in. deep. The square gaps between the bars quickly fill with crushed rock, and it is this which sustains a large proportion of the wear.

Mr. K. S. W. Orr asked (p. 672) for further details of the relative track costs for bulldozers employed on the two sites. The comparative costs given in the paper referred to similar models. The dozers and rippers are used for preparing site roads, levelling the carpet for the draglines, clearing the spillage at loading points and levelling subsoil and topsoil, etc. No attempt is made to use these machines for removing rock overburden since the haul distances are far too great.

At Acorn Bank some dozers were operated for 20 000 h before replacement. At Westfield the first machines were replaced after 15 000 h.

Since the daily average use of machines with rippers is only one hour, on light ripping, the difference in track costs has not been recorded.

Mr. B. J. Greenland's and Mr. G. J. Shrimpton's questions (pp. 673 and 675) on the relative costs of slurries and AN–FO mixes are perhaps best answered by the following, which shows the drilling and blasting requirements for rock of the type overlying the coal at Westfield.

	AN–FO	*Cosminex*
Drilling machine	58 BH Joy	58 BH Joy
Hole diameter	$6\frac{1}{4}$ in.	$6\frac{1}{4}$ in.
Spacing	12 ft by 12 ft	18 ft by 18 ft
Depth of hole (includes 5 ft overdrilling)	35 ft	35 ft
Yield per hole	160 yd^3	360 yd^3
Explosive required	194 lb	145 lb
Yield per lb	0·87 yd^3	2·5 yd^3

Naturally, the drilling and blasting details vary considerably from the above when working in the coal seams and the partings.

At present, at Westfield approximately 26 000 lb of Cosminex slurry per week are produced. At this output 1 lb of slurry costs about twice as much as 1 lb of AN–FO filled in sealed tin cans.

At this output slurry is just viable but, of course, it must be understood that slurry can rarely be competitive with AN–FO on a dry site where canning is not required, unless drilling costs are exceptionally high.

Mr. E. S. Chaplin's remarks (p. 674) regarding the allocation of dump trucks concern a problem currently receiving a great deal of thought. The control of dump trucks appears to be something which a computer could handle satisfactorily and the design of a special computer is in hand. The author's original thoughts were centred on an analogue computer, but specialist advice now seems to favour a digital computer as being more reliable.

In reply to Mr. M. W. Rushton (pp. 674–5), at Westfield two sizes of shovel were thought desirable—the larger machines for the bigger faces, the smaller machines for the thin seams. Since it was thought preferable to make the dump trucks interchangeable, the size of dump truck was a compromise.

Generally, a useful rule is that the shovel should be able to fill the dump truck in from two to five passes. To fill a dump truck in one pass, as was sometimes done at Acorn Bank when filling direct from the 25-yd^3 dragline into an 18-yd^3 dump truck, strains the vehicle considerably. Loading cycles in excess of five passes are only economic when long haul times are involved.

Dr. W. Tilmann: In reply to the question raised by Mr. R. O. Morris (p. 675), freezing of the groundwater along the margins of open-cuts has been examined thoroughly. The thickness of the water-carrying layers necessitates extraordinarily thick walls and the deep and large open-cuts would require great wall lengths. This entails a huge consumption of electricity for establishing and maintaining the freezing walls. Since the safety factor of such a wall is unknown, the entire method was regarded as unsatisfactory.

In reply to Mr. N. R. Cameron (p. 675), the cutting force of the dredges is great enough to win coal containing a large amount of fossil wood, even when occurring as big chunks. The tree roots on the terrain surface will, however, be removed by special means, if necessary by blasting, because these roots would hamper the flow of overburden on the dredges as well as on the succeeding belt conveyor lines.

Mr. W. B. Bell and Dr. P. E. O'Sullivan: In reply to the question by Mr. T. T. Heywood (p. 670), similar experiences to his have been noted. Of recent years, however, rough service lamps which contain more filament supports than in the normal G.L.S. lamps are obtainable. These offset to some extent breakage due to traction effects. Where breakage still occurs, even when such lamps are used, a higher voltage lamp can be employed, these having the effect of providing a stronger filament at the lower mains voltage.

In reply to Mr. D. E. Speight (p. 670), on the sites visited no discharge lamps were employed. However, with respect to such discharge lamps it would be beneficial as far as colour is concerned if the mercury bulb fluorescent coated lamps were used. These lamps effect a good colour difference between coal and the associated stone and are also between four and five times as efficient as the normal G.L.S. metal filament lamp. A new source which has only recently come on to the market is the tungsten iodine source—a tubular tungsten lamp of small dimensions having an efficiency of twice the G.L.S. lamp and a much improved life. Where colour discrimination is an essential, the low-pressure sodium vapour lamp, which has an efficiency of between 120–150 lumens per watt (i.e. ten times that of the G.L.S. lamp) and a guaranteed life of 6000 h can be used with considerable success.

To answer the question by Mr. A. Major-Stevenson (p. 670), 24-V lamps are frequently employed for very heavy and arduous duties associated with Caterpillar tractors and machinery where vibration is very marked. Such equipment normally limits its current, and therefore its wattage, to below half of that queried by Mr. Major-Stevenson. These lamps are normally housed in small totally enclosed lamp housings (i.e. headlamps, etc.). When higher wattages are employed heat dissipation would therefore have to be considered and possibly the lamp housing should be increased in size. No data are available for such wattage lamps and, in fact, no manufacturer has yet considered making this wattage of lamp.

GROUP IV—OPENCAST MINING
(bedded deposits and general questions)

Session 9—Operations (Papers 31–34)

Chairman: Mr. R. F. Lethbridge *Rapporteur:* Mr. P. Best

622.271.3:622.349.52(943.4)

Paper 31

Selective Mining Practice at Mary Kathleen Uranium Mine, Queensland

D. L. MUNRO, B.Sc. (Geology)

Former Chief Mine Geologist, Mary Kathleen Uranium, Ltd.

SYNOPSIS

Mary Kathleen is an opencast uranium mine in northwest Queensland, Australia. The orebody was discovered in July 1954 and a contract to supply 4500 short tons of uranium oxide to the United Kingdom Atomic Energy Authority was formally completed in December 1963. The mine and township have recently been placed on a care and maintenance basis.

The opencast has an average daily production of 5000 short tons per day from eight benches within a restricted surface area. During the contract some 7 800 000 short tons of material have been mined, of which 2 900 000 tons was ore.

The very irregular ore occurrence of a valuable mineral has justified the employment of a highly selective mining operation. Sampling of all blast-holes is carried out using a geiger probe and each face exposure is geologically mapped. Comparison between the geological mapping of the bench face and the probe profile from the blast-holes behind the face is used as the basis for selective mining. Based on this comparison, firing instructions are issued by the geologist to the mine foreman. Holes are fired in the manner best calculated to separate the ore from the waste, with only secondary attention paid to the effect on primary breaking.

The uranium deposit of Mary Kathleen is situated in northwest Queensland, Australia, approximately halfway between the townships of Mt. Isa and Cloncurry, at approx. 140° 00′ 45″ E and 20° 44′ 50″ S.

The deposit lies within the rugged Argylla Ranges which comprise a series of mainly N–S trending hills and ridges rising up to 600 ft above the intervening valley floors. The uniform level of the hilltops at present about 1600 ft above sea level suggests that a pre-erosion peneplain existed in the area. The orebody was originally seen to outcrop on a westerly facing hillside with slopes of up to 40°, the outcrop being exposed to a height of 250 ft above the valley floor.

The average rainfall for the area is 17 in., there are no perennial streams, and most of the rain falls in isolated storms during the summer months of November to March. The vegetation consists mainly of stunted eucalyptus, acacia and spinifex. The average maximum temperature ranges from 76·7°F in July to 99·7°F in December. The average minimum temperature in July is 51·5°F and the relative humidity is

27 per cent in both months. The average evaporation rate is 108 in. per annum.

The deposit was discovered by prospectors of the Walton McConachy syndicate at the beginning of July 1954, and was the climax of a very intensive ground search carried out by hundreds of part-time prospectors from Mt. Isa during the uranium boom from March to July 1954. Numerous small deposits, mostly in the vicinity of Mt. Isa, were discovered in various rock environments, but Mary Kathleen is the only large deposit and, apart from a few tons of hand-sorted material from the Milo lease, is the only deposit that has been worked.

Proving of the deposit was completed by June 1956. From June 1956 to June 1958 the company and its contractors built a dam with capacity for three years' water supply, eight miles of pipeline, treatment plant and a modern township of 250 houses complete with most amenities for a population of 1100. The township is situated in a pleasant valley four miles from the treatment plant and is connected to Cloncurry and Mt. Isa by the part-bitumen, part-earth formed northwest highway. The company generates its own power supply and the workshops are reasonably self-sufficient and able to undertake all but major fabrication.

The flowsheet of the treatment plant includes a fine-grinding and acid-leach circuit followed by a fixed-resin-bed ion exchange and magnesia precipitation of the uranium values. Sulphuric acid is manufactured on site from elemental sulphur in a Simon Carves Monsanto contact plant with a nominal capacity of 110 tons of 100 per cent acid per day.

Run-of-mine ore is delivered by Euclid rear dumps to a 20-ft by 60-in Amsco apron feeder feeding directly into an Allis Chalmers 60-in by 48-in jaw crusher set at 4½-in closed setting. The 3-in undersize is delivered directly to the fine-crushing section while the oversize +3 in is delivered to the surge bin of the electronic sorter. This is a device which makes use of the radioactive nature of the ore to separate the ore from the waste rock.

GEOLOGY

The Mary Kathleen deposit occurs within a portion of the Australian Precambrian Shield in highly altered metasediments of the Corella Formation of Lower Proterozoic age. The orebody is located in the axial zone of a syncline. Granite occurs in elongated belts to the east and the west, and the intervening Corella beds, which vary from quartzites, lime–silicate granulites, limestones and stressed basic rocks, have been subjected to waves of alteration, injection and replacement. Quartz–feldspar porphyry and dolorite dykes, and locally intense replacement by garnet, are superimposed on an area that had already been folded and faulted and regionally scapolitized.

Garnetization is an important phenomenon in that all the uranium mineralization in the immediate vicinity is associated with selective replacement of the intensely garnetized zones by rare-earth and uranium minerals. Garnet replacement, in areas removed from the orebody when primary rock structures have not been obliterated, is seen to be controlled by both bedding and cross-joints and shears. In the ore zone selective replacement of intensely garnetized rocks by allanite with uraninite appears to be largely controlled by the pre-existing joints, of which the fairly flat-lying system is well developed, and is apparently the dominant ore control.

Replacement in the orebody has occurred in a belt of garnet–diopside–apatite granulite by the development of black allanite and brown stillwellite (a new rare-earth borosilicate) with which primary uraninite is associated in fine dispersion. Sulphides of iron and copper, carrying small amounts of cobalt and nickel and other base metals, occur both disseminated through the orebody and in massive form associated with faults and shear zones. To within a depth of about 100 ft from the surface, oxidation of the uranium mineral is partial or complete, with the formation of gummite and uranophane, either as haloes around the uraninite grains or re-deposited on joint surfaces.

The orebody is bounded both to the east and west by strong north–south shears. Neither of these has affected the opencut design or appears likely to affect the stability of the walls of the opencut.

PROVING OF THE DEPOSIT

Initial testing of the deposit was done by diamond drilling with vertical and inclined holes, mostly 300–500 ft deep, collared on a grid pattern, the driving of an exploratory adit, costeaning and the examination of surface exposures. The ore occurrences varied in thickness from a few inches to 150 ft and are best pictured as an irregular honeycomb of connected shoots of mineralization separated by waste blocks. The main mineralized zone is contained within a 600-ft cube and has a waste to ore ratio of 1:1.

Throughout the testing programme full advantage was taken of the radioactive nature of the orebody. The use of various geiger instruments allowed the rapid, positive and inexpensive solution to many problems. Surface radiometric readings were taken on a close-set grid over the mineralized zone and the resultant radiometric contouring clearly showed the boundary and extent of the occurrence. It did not, however, clarify the internal structure. The diamond-drill core was monitored with a hand-held Phillips PW4010 geiger counter, a lightweight insensitive instrument ideally suited to evaluation, and the readings, taken at 6-in intervals, were presented graphically and served as the basis for splitting the core for later assay. In addition, every hole was probed.

A geiger counter, protected by a steel sheath and connected by cable to a recording meter, was lowered down the hole and readings were taken at 1-ft intervals. The result was again shown graphically.

The two graphs provided a check on each other. The probe graph provided a sure picture of the overall radioactivity in the hole, and was of particular usefulness in reaching a decision whether to stop or continue a hole; it was also used to calculate grade in sections of low core recovery. The bulk of the assays of diamond-drill core were carried out cheaply and accurately by radiometric assay, the equipment being calibrated against chemical standards and a range of chemical checks being carried out on selected samples.

OPENCUT DESIGN

Because of the irregular ore occurrence selective mining by underground methods was considered impractical. This has since been confirmed by the detailed geological mapping of the opencut advance. The hillside exposure of the orebody was, however, ideally suited to an opencut, but the irregular occurrence of valuable mineral dictated that the normal opencast approach of large equipment with large tonnages for low cost would have to be subordinated to the need for selective mining. It was decided, therefore, to mine from shallow 25-ft high benches and use equipment of relatively small capacity but greater manoeuvrability.

The present excavation design provides for the development of benches into the original hill slope to a total height of 350 ft above the valley floor and to a final overall batter of 50°. Below the valley floor a spiral entry opencut is planned to an ultimate depth of 200 ft. In the final position two 25-ft benches are taken back to form one 50-ft 67° face. The design was based on providing the contract requirement of 4500 short tons U_3O_8 plus a generous amount for contingencies. At the completion of the contract the top ten benches to a depth of 250 ft were back to their design limit and the cut was down to reduced level 925 or 75 ft beneath the valley floor.

The excavation was designed to remove some 12 000 000 short tons of material, of which 4 000 000 tons was ore and the remainder waste rock. Of this 1 400 000 tons was waste material outside the ore limits and was classed as 'overburden'. This was removed as a separate operation at relatively low cost using a $2\frac{1}{2}$-yd^3 54 Ruston-Bucyrus shovel loading three Euclids on the two shifts. During the years 1959–1961 the annual production rate was 1 500 000 short tons, of which 500 000 tons was ore, 500 000 tons internal waste and 500 000 tons 'overburden'. At the completion of the contract approximately 7 800 000 short tons of material had been removed, of which 2 900 000 tons was ore, 3 500 000 tons was internal waste and the remaining

1 400 000 tons was 'overburden'. The waste to ore ratio had therefore been better than expected.

DRILLING AND BLASTING

All primary drilling on the ore benches is carried out by three mobile Ingersoll-Rand CM2 Crawl-Ir machines using 10-ft by $1\frac{1}{2}$-in diameter round Coromant extension drill-rods and $2\frac{3}{4}$-in diameter tungsten carbide bits. The Crawl-Ir units work six shifts per day drilling an average of 180 ft per machine shift in down- and toe-holes. Down-holes are drilled a single row at a time with 8-ft spacing and 6-ft burden giving a factor of 5 short tons broken per foot drilled. Bench down-holes are normally drilled 2–3 ft below the bench floor. Average performances are 600 ft per bit, 600 ft per rod and 350 ft per coupling.

Drilling in overburden was carried out using an Ingersoll-Rand 'Drillmaster' fitted with a down-the-hole 'Depthmaster' drill and 6-in diameter tungsten carbide bit. Holes were drilled on a pattern of 15-ft burden and 15-ft spacing, giving a factor of 25 short tons broken per foot drilled. The average footage drilled per shift was 100 ft, the average footage drilled per bit was 1500 ft and the bits were sharpened every 30 ft.

Increasing use of ammonium nitrate–fuel oil and slurry mixtures has been made in primary blasting over the last four years. AN–molasses slurry with 10 per cent standard explosive has been used for the overburden and AN–FO with 20 per cent standard explosive for the $2\frac{3}{4}$-in holes on the ore benches. Initiation in both cases is by safety fuse connected to Cordtex detonating fuse. In both cases approximately 4 tons are broken for each pound of explosive. The AN explosive is mixed at the charging site using a hand-operated mixer, the bowl of which has been lined with an Epikote–D.E.T. coating. The restrictions imposed by selective mining preclude pneumatic loading. Secondary breaking is carried out mainly by plastering using an AN–FO mixture with standard primer and Cordtex of 7-lb weight in a plastic bag.

LOADING AND HAULAGE

Five shovel shifts are worked each day on opencut production and stockpile loading. The loading equipment comprises one $2\frac{1}{2}$-yd^3 54 RB Ward-Leonard electric shovel, three $1\frac{1}{2}$-yd^3 38 RB electric shovels, and three $1\frac{1}{2}$-yd^3 38 RB diesel shovels. Loading statistics for the equipment are: for the 54 RB, present output per shovel shift, 1250 short tons, maximum output per shovel shift, 2000 short tons; and for the 38 RB, present output per shovel shift, 985 short tons; and maximum output per shovel shift, 1200 short tons.

All tonnage from the opencut is hauled in B6FD 9·7-yd^3 rear-dump Euclid trucks, these units carrying an average load of 19 tons. Nine trucks are worked on day shift and six on afternoon shift, the average length of haul from the opencut being 4000 ft and the average tonnage per Euclid shift 420 short tons. Three D8 Caterpillar bulldozers are employed squaring up rock piles and cleaning benches and stockpiles. Gravel-surfaced haulage roads are graded each weekend and resurfaced periodically. A water-tanker is in constant operation each shift watering all benches and roads in use to lay dust.

The average mining department establishment consisted of 18 staff members and a labour force of 60 men working two shifts a day for a 5-day week. During the 'overburden' programme the production performance was 82 tons/man-shift.

SURVEY AND PROBING

All benches are identified by their floor elevation with reference to an arbitrarily selected base line datum and all positions in the opencut are referred to a co-ordinate system based on a N–S base line established along the original lease boundary. All vertical blast-holes for primary breaking have to carry the dual function of breaking ground and providing a sample of the ground intersected. Equal weight is given to both functions. All vertical blast-holes are therefore surveyed and probed as a matter of daily routine, and each hole is identified by a numbered plate which remains to identify the hole until such time as the hole is fired.

Sampling is carried out by probing the down-holes with a geiger probe. This is done concurrently with the survey using the identification established by the surveyor. Readings taken at 1-ft intervals as the probe is raised are manually recorded and later divided into significant intersections of ore or waste, depending on whether the value is above or below the cutoff grade. This survey and probe information is collated on a series of assay plans and longitudinal sections. These probe sections provide a view of the ore attitude and grade behind the face and are used in conjunction with the geological face mapping to direct the selective firings. The detailed sampling obtained by the probe also provides a method of predicting tonnage and grade (this is discussed later).

GEOLOGICAL BENCH MAPPING

Geological mapping at Mary Kathleen is an essential part of the selective mining operation and is therefore carried out as part of the daily routine. The principal aid to rapid geological mapping is the prior monitoring and marking of the bench faces. Beginning as soon as practicable after a firing so as to reach the top of the face by standing on the fired rill, the faces are first scaled down, then hosed to give the clearest view, monitored with a Phillips hand-held geiger counter, and the outline of the ore-shoot is traced out in chalk by the staff sampler, who uses both the geiger counter and his eye to delineate the ore from the waste. The sampler follows closely behind painting the chalk lines white and painting the letters A and B to identify ore and the letter W to designate waste.

This information is transferred by the geologist to a prepared mapping sheet which shows the surveyed position of relevant reference points in plan and longitudinal section. Using this framework the geologist sketches the ore outline, significant fractures and other geological information in longitudinal section and then projects to plan.

SELECTIVE MINING

Primary Selective Firing

The immediate purpose of both the probe and geological information is to plan selective firings. Normal firings are usually from 1000 to 2000 tons and the position and nature of the firings depend on the state of the benches, the shovel availability and the mill requirements. From his detailed knowledge of the mine operation and his visual estimate of the bench faces the mine foreman selects a certain group of holes to provide him with an ore or a waste firing and contacts the geologist to have this firing confirmed.

All down-holes are fired only after the issuing of firing instructions signed by the geologist, which are passed from the mine foreman through the shift boss to the powder monkey. The firing instruction identifies the bench, section, hole numbers and depths to which individual holes are to be fired. (This information is later passed through the various survey records.) Decision on waste firings is quickly taken, while decision on ore firings is arrived at by comparison of the longitudinal sections obtained from the geological mapping and the probe profile and is frequently very complex.

Selective firing was first begun in 1957, but was not systematically and rigorously applied until May 1959. The firings first attempted were very complicated, but have gradually been simplified over the last four years. In the first year attempts were made to carve the face to fit the ore occurrence and holes were fired to uneven depths, and in some cases only the top 10 ft was fired, the remainder being filled with stemming. These uneven holes, and especially the short holes, defeated their very objective as the ore (or waste, as the case might be) at the top of the face did not come out but hung up. When brought down by later toe-holes this caused the same dilution as if the holes had been fired full depth. Over the last few years the rule has been never to fire less than 15 ft and always to rationalize adjacent holes to fire to the same depth. The practice of leaving gaps in a string of holes has been largely discontinued. The principal exception to these rules is the case where the ore is controlled by a strong obvious plane of weakness and holes are fired to this plane. The more satisfactory condition is to fire ore off the top, leaving the waste in the toe. In this condition the holes are fired to a depth 2 ft beneath the ore outline.

Firings are identified as ore, waste or mixture firings, the latter being most common. The probe section may match the bench mapping or may be the mirror image of it; or in very bad cases the ore may be irregularly disposed in both face and probe sections. As the firing is dictated by a very variable mining programme, there is little point in too much forward planning.

Muckpile Marking and Sorting

Marking and sorting of the muckpile are very necessary adjuncts to selective firing, their effectiveness being directly related to the amount of time which can be devoted to them. This is dependent on the number of available benches, the degree of access to these benches and the required output to the primary crusher. In the earlier years, with lower ore production of 2000 short tons of ore per day, the bench sorting was more efficient than at the maximum production rate of 3200 short tons of ore per day in March and April 1963.

The full sorting cycle begins with the dozing of the primary break which, except in special circumstances, must be done before the rock pile is marked. If done after the marking, all the work is obliterated. Marking of the rock pile is done by the chief sampler and an assistant. Using a modified Phillips PW4010 geiger counter the chief sampler monitors the large lumps in the firing, generally marking all waste lumps in chalk with a white cross, which is painted by the sampler. Under most conditions it is preferable to remove the waste from the ore, it being seldom possible to remove all the ore (as it fractures more easily), and thus the pile may still go as ore. Where, however, there are a few lumps of ore in a predominantly waste firing, the ore is marked with a red dot and removed. The character of each pile is thus identified for the supervisory staff who direct the following sorting operations.

The next step is the removal of all waste pieces within reach by a 5-ton rubber-tyred hydraulic mobile crane. Two men per shift are employed on this work, comprising one crane driver and one labourer who slings the lumps with a rope sling or nip chain for attachment to the crane hook. Only a certain proportion of each pile can be reached by the crane from any particular side and access to both sides of a rock pile is therefore important. Even with access from both sides the crane can only remove a certain proportion and the heap must then be shovelled and loaded. Under low-volume production conditions the shovel operation can play a very important part in improving selectivity and reducing dilution. The shovel operator can first place all large lumps to one side and, as the waste generally breaks larger than the ore, these rejected lumps are predominantly waste. Secondly, after a limited shovelling time in one heap the shovel can be withdrawn from that particular pile and the pile re-marked and re-sorted by the 5-ton mobile crane. With a production rate of 3200 short tons/day and restricted bench area it was not found possible to perform the shovel portion of the sorting cycle and the crane sorting was even curtailed or omitted if the ore was needed at the primary crusher.

The operation with the crane appears crude and primitive, but no more effective method has yet been devised to cope with the work as no positive-acting rock-grapple machine appears to be commercially available with the required combination of lifting capacity, reach and mobility. Working two shifts, the crane is estimated to remove an average of 150–200 tons/day of waste which would otherwise go to the mill to be crushed and electronically sorted. This means that an additional 50 000 short tons of payable material can be sent to the crusher each year.

Secondary Breaking

Wholesale secondary breaking improves the shovel loading and the primary crusher feed, but is deleterious to the selective mining. Properly organized, the secondary breaking should take its place in the sorting cycle, depending on the condition of the muckpile, and is specifically directed by the shift boss as conditions change. If the firing is a mixture and the waste lumps are too large for the crane, then those waste lumps should be plastered, only sufficient to break them for the crane. Large ore lumps should not be plastered until the relevant portion of the heap has been properly sorted by the crane. Ore and waste should not be plastered together. If the firing is dominantly ore then large waste lumps should be left unplastered so that they will be rejected by the shovel operator. After successive stages of the sorting cycle the firing should be examined by the shift boss.

FINAL MONITORING AND DISCRIMINATION

Assessment of Euclid Loads

All Euclid truck loads, as they leave the opencut, pass through a combined weighbridge and radiometric grade assessment 'discriminator'. Here the load is scanned by 6 geiger tubes mounted in a framework in the form of an inverted parabola. These tubes scan the top portion of the load, which is taken as representative of the whole load. Impulses from the tubes are integrated and the result is read off a simple meter dial. The loaded vehicle is then directed by the discriminator operator to its destination—to ore or to waste or to either of the two grades of marginal material which, for the time being, are being stockpiled. The discriminator operator keeps a record of all material passing through the station, listing, for every truck load, the truck identification number, its net weight, discriminator grade, the source of the material (bench and grid section number) and its destination. He has radio-telephone communication with the crusher and with the vehicles of the opencut supervisors, who advise him of any shovel transfer from one bench section to another. Radio-telephones also enable the foreman and shift boss, at any time, to enquire from the discriminator operator how the grade from any particular bench is running. The completed discriminator sheets form the basis of most of the production records of mining operations.

Calibration

Calibration of the discriminator is carried out by passing crude ore from the opencut in normal Euclid loads through the discriminator to the mill primary crusher, then through both secondary crushers to be drawn off as $-\frac{1}{2}$-in crushed material which returns through the discriminator. The forward crude ore load through the discriminator is weighed and the discriminator reading taken. The return $-\frac{1}{2}$-in crushed load is weighed, discriminator reading taken and a 10-lb sample cut in a series of runs from the back of the Euclid tray. If it is required as a check a 'mill sample' can be taken at the mill discharge point above the fine-ore bins. With the co-operation of the mill operators it is possible to feed the loads through, with a short delay between each, so that the $-\frac{1}{2}$-in return crushed load almost corresponds to the forward crude load. The mass and shape of the returning sample should also be as similar as possible to the forward load. Monthly reconciliations between opencut production and mill head show the discriminator to be a practical bulk-sampling device and confirm that the calibration is valid.

EVALUATION OF RESULTS

It is difficult to determine the effectiveness of the selective mining as there are no absolute standards against which to measure the mining performance. Opencut production figures are reconciled to the mill head values and corrections passed back to the opencut figures which are then accepted as correct. The production figures are then compared with the ore reserve estimate based on the diamond drilling results and with the probe estimate which is derived from the detailed probe sampling. The comparisons between these estimates and production have to serve the contradictory purpose of assessing performance and improving the estimates themselves. It can, however, be stated as a personal observation that the 6-month period of worst result with low grade and maximum dilution coincided with a time when selective mining procedures were relaxed. The accent on selective mining makes all the staff grade-conscious.

Detailed comparisons are only available for the years 1959–1963 inclusive (Tables I and II). These show the ore reserve calculations to be very conservative. Most of the error is believed to be in the upper benches which were mined in 1959–1960 and which have now reached their design limit.

TABLE I.—*Years 1959–1963*

Details	Ore			Waste	Ore and Waste
	Short tons	Grade lb/ton	Content lb U_3O_8	Short tons	Short tons
Production	2 444 577	3·57	8 738 144	4 194 434	6 639 011
Ore reserve	1 402 136	3·86	5 417 540	5 006 864	6 409 000
Probe	1 821 469	5·90	10 759 999	4 106 794	5 928 263

TABLE II.—*Years 1961–1963*

Details	Ore			Waste	Ore and Waste
	Short tons	Grade lb/ton	Content lb U_3O_8	Short tons	Short tons
Production	1 539 959	3·63	5 583 599	2 059 527	3 599 486
Ore reserve	1 307 901	4·11	5 378 350	2 141 499	3 449 400
Probe	1 123 785	6·19	6 954 394	2 415 989	3 539 774

Detailed comparisons between probe profiles and the same bench face revealed after firing have confirmed that the probe gives an accurate measure of width of ore intersection, but the assessment of grade by the probe is still uncertain. The tonnage figures derived from the probe are therefore taken as the ideal or standard. The difference between the probe and production figures can therefore be taken as an expression of mining dilution. This has varied from 34 to 37 per cent over the last five years of production. The ore reserve figures contain an allowance for mining dilution. The disparity between the ore reserve and production figures can be taken as an expression of the inadequacy of the original diamond-drilling information upon which reliably to base an ore reserve assessment over a relatively small amount of the orebody.

The purpose of the statistical information recorded is to improve the prediction for both the immediate mining programme and for the total remaining ore reserve. Factors used for short-term prediction have been derived from (1) ore reserve estimates, (2) previous production records, and (3) probe records. The results of application of the first two have been disappointing. Reference to the previous tabulations shows that there is a large difference between probe estimate and production result, but detailed examination of a number of comparisons showed that the differences were generally consistent. The progressive figures for the four years 1959–1962 showed that applying correction factors of +33 per cent to the probe ore and −19 per cent to the probe content gave estimates very close to the actual production. This method of prediction was used in the last firings of the contract with very satisfactory results. More experience is needed, but the method does appear to work even for individual firings. It also has the great advantage that it represents a sample of the orebody which is already drilled, and is therefore closer to the probable mining advance.

The value of the product is the guiding factor that has dictated the development of a highly selective mining operation for this very irregular mineral occurrence. Economic considerations were the motivating force behind the selective mining and future economic conditions will decide how these procedures are adapted and modified.

Acknowledgements.—In writing the paper the author has drawn freely upon various company publications and unpublished reports and in particular upon those of two former mine superintendents at Mary Kathleen—B. C. Ryan and L. R. Baster. The opinions stated are, however, those of the author. Thanks are expressed to the Directors of Mary Kathleen Uranium, Ltd., Conzinc Riotinto of Australia, Ltd., and the former manager of operations at Mary Kathleen for permission to publish this paper.

622.271.4:622.349.21(881.2) *Paper 32*

Stripping Operations of the Demerara Bauxite Co., Ltd., British Guiana

WILLIAM FORBES, B.Sc.(Min. Eng.)
General Superintendent (Bauxite Division) Demerara Bauxite Co., Ltd., Mackenzie,
British Guiana, South America

SYNOPSIS

The bauxite deposits of the Demerara Bauxite Co., Ltd., in British Guiana, are covered by a variable, but usually considerable, thickness of overburden, which must be removed before the ore can be mined. The stripping operations, employing hydraulic monitors, dredge pumps, draglines, and bucket-wheel excavators, are described, as well as the local conditions encountered and the types of overburden materials handled. A summary of the development of the Company's stripping methods, from the hand labour of 1917 to the present-day use of large modern equipment, is presented. The methods mentioned above are discussed and their comparative merits assessed in the light of local experience. The newer bucket-wheel excavator system at present in use is described in more detail than the older, more widely known, hydraulic and dragline stripping methods.

The Demerara Bauxite Co., Ltd., known as Demba, is a fully owned subsidiary of the Aluminum Company of Canada which, in turn, is the principal operating subsidiary of Aluminium Limited. Demba is engaged in mining and processing bauxite, and producing alumina, from its properties in British Guiana. Exports of metallurgical, abrasive, refractory, and chemical-grade bauxite, as well as alumina, are made to about 30 countries. The operations are centred in the town of Mackenzie situated on the Demerara River some 65 miles from its mouth, where Georgetown, the chief port and capital city of the country, is located. With the exception of Mackenzie and the towns of Wismar and Christianburg on the opposite bank of the Demerara River, the area is almost completely uninhabited and undeveloped; but Mackenzie itself is well laid out with paved roads and air-conditioned shops, a 100-bed hospital, primary, secondary and trade schools, and all the amenities of a modern community. There are about 22 000 inhabitants in the area, about 7000 of whom live in Mackenzie.

The mining operations are in two main areas, one 5 to 10 miles south and the other 38 miles southeast of Mackenzie, to which they are connected by narrow-gauge railroad. The processing works consists of the bauxite plant with its own dock and two ship loaders, and the alumina plant with a dock and ship loader. Metallurgical-grade bauxite can be loaded to ships at the rate of 1100 long tons per hour, alumina and calcined bauxite at 600. Ships can usually only be loaded to about 19-ft draught and bauxite ships are therefore 'topped up' in Trinidad where a deep-water ore-transhipping station is maintained; alumina ships are 'topped up' in Jamaica by an associated company, Alcan Jamaica, Ltd. Demba's total investment in plant and facilities at Mackenzie amounts to about £25 000 000, which supports a payroll of about 3500 people.

GENERAL

The mining area consists generally of a white sand plateau deeply incised by major streams tributary to the Demerara and Berbice Rivers and covered by savannah forest, except in the creek valleys which support tropical rain forest. To the south of the mining areas there are extensive open savannahs.

The bauxite outcrops near Mackenzie were first reported in 1868, but not recognized as an ore of aluminium. They were described as being rich in aluminium hydroxide by Sir John Harrison, the Government Geologist, in 1898, but there was little interest in aluminium ores at that time. It was not until 1914 that mining of the Demerara bauxites was seriously considered, owing mainly to war conditions and the resultant increased consumption of aluminium. Demba was incorporated in 1916, undertaking its first mining operations in 1917, and mine production over the years has increased from less than 3000 long tons in 1917 to the present 3 500 000 long tons per year. Roughly 3 tons of bauxite, as mined, are required to produce 1 ton of alumina or calcined bauxite and about 1½ tons to produce 1 ton of metallurgical-grade bauxite.

Production is from opencast mines, the overburden consisting of sediments of the Berbice Formation, a continental deltaic deposit laid down in shallow water. Lithologically, this formation consists of white unconsolidated quartz sands with indurated sand patches, sandy clays, and clays with intercalated bands of sandy clay. The indurated sands encountered are usually thin, iron-impregnated horizons. The surface of this formation forms a plateau with an immature drainage pattern and a relatively slight runoff, since the rainfall percolates through the sand to the underlying clay layers. Up to 200 ft of overburden is at present being removed before mining, but bauxite has been encountered in drill-holes at depths of over 300 ft.

Stripping is mainly by hydraulicking combined with dredge pumps, draglines and bucket-wheel excavators; the stripping difficulties arise from the depth of overburden to be removed and the effect of tropical rainfall on the stripping faces and spoil piles. The annual rainfall, averaging over 100 in., is not excessive for the tropics, but occurs mainly in one short season in which torrential downpours may cause spoil piles to slump and bury drainage ditches and railway tracks at a time when the drainage system is already taxed to the limit. From the stripping surface the unconsolidated sands allow the percolation of rainwater to the clay horizon where it emerges, generally, in the form of small swift springs flowing from the dragline stripping faces; where there is no clay between the bauxite and sand, the springs occur at the contact between the sand and bauxite. Small deltas of sand are thus deposited on top of the bauxite and must be cleared away before mining begins. The clays, which are usually found immediately above the bauxite horizon, are normally firm and easily handled but, as they have a high moisture content, are liable to spall from the dragline face, sometimes making operating conditions for dragline equipment very treacherous and adding to the clean-up work required in the pit below.

Although overburden removal operations are harassed by various factors associated with tropical conditions, it is not necessary to contend with some of the complications encountered in more populated or arable areas. The boundaries of mining areas are not limited by industrial or agricultural zones, highways or railways, and the extent of property leases is sufficient to accommodate the dump areas. It is doubtful whether stripping, to the depths necessary to uncover the bauxite, would be an economic proposition in more populous and developed areas, where these advantages of adequate space would be unlikely to exist.

DISCUSSION OF DEVELOPMENT OF STRIPPING METHODS

Stripping methods used at Mackenzie have progressed from the hand loading of mule carts in 1917 to the present monitors and dredge pumps, walking draglines and bucket-wheel excavators. Only the outcrops were worked in the early years, but mining quickly advanced into increasingly heavy overburden. During the war years all the orebodies covered by light overburden were worked out to meet heavy wartime demands. By 1945, tractor-drawn and self-propelled scrapers were removing up to 40 ft of clay to uncover bauxite. This was the economic limit of these units under the prevailing conditions. This economic stripping limit has now been increased to 200 ft, despite greatly increased costs for labour and supplies and a fairly constant price for aluminium and calcined bauxite, mainly through a very heavy investment in large stripping equipment of modern design. The earliest mechanized stripping was by small power shovels and draglines loading into dump cars, trammed first by hand and later, as the operations grew larger, by locomotives; then by power-shovel loading to track-mounted wagons hauled by tractors. In the late 1930s these, in turn, were superseded by tractor scrapers and hydraulic stripping. From the late 1930s until the early 1950s tractor- or self-propelled scrapers uncovered most of the ore; hydraulic stripping continued, but did not actually uncover bauxite. Various other methods were tried during this period, such as drag scrapers and excavators loading on to a conveyor-belt system, but the results were discouraging.

Tractor-drawn and self-propelled scraper units were adequate up to about 1945, but were obviously going to be unable to cope with the haulage distances and the quantity of overburden to be removed in the 1950s, unless the use of an extremely large fleet were contemplated. This would have been very costly unless some reduction in the high operating and maintenance costs could have been achieved; the economic operation of a tractor-scraper fleet, as is well known, requires a high level of mechanical and operational supervision. The tractor- or self-propelled scraper removal of 40 ft of tough slippery clay, at a high rate, was not a very attractive proposition. The problem of spoil disposal would have become critical with the increase in overburden thickness; disposal by tractor-scrapers in the mined-out area would have been impractical because of interference with the railroad mining operations; disposal beyond the limits of mining would have required hauls of up to half a mile. A further serious objection to tractor-scraper or rubber-tyred stripping units was the sharp decrease in output during the wet seasons.

Hydraulicking was first introduced in 1938 and continued until 1952, when it was combined with sand pumping. Hydraulicking, supplemented by the use of ejectors, was also used in the same period for a relatively short time. Overburden removal by large draglines began in 1952 with the purchase of one dragline, and it has been increased until there are now four such machines operating. In 1955 a cutter-head suction dredge was introduced as an experimental method of overburden removal; the costs were higher than sand pumping costs and it was taken out of service after two years. A bucket-wheel excavator, with about $1\frac{1}{2}$ miles of conveyor system, was commissioned in 1962 to replace the combination of hydraulicking and sand pumping in the Arrowcane mine, where sand pumping had become unattractive owing to the distance to the sand-pump discharge area and the necessity of having several sand pumps in the face to handle the volume of overburden present. Two bucket-wheel excavator units are expected to be in operation by the end of 1964, as well as a small reclaiming-type bucket-wheel excavator, which will be used as a mobile stripper.

It should be mentioned in passing that it is not practical to use underground mining methods to recover the bauxite in the Mackenzie area as both the back and floor of the deposits consist of a wet clayey material, completely incompetent. A test drift was actually driven in the bauxite in 1937, but was quickly abandoned due to the low bauxite recovery and high timbering cost.

At present, sand and clay overburden is being stripped off at the rate of 12 000 000 bank yd^3 per year to uncover 3 500 000 long tons of bauxite and the overburden-to-ore ratios are increasing (Fig. 1). The investment in stripping equipment, not including miscellaneous tractors, etc., totals £2 700 000. In addition to normal replacements, further large stripping investment will be necessary by 1970.

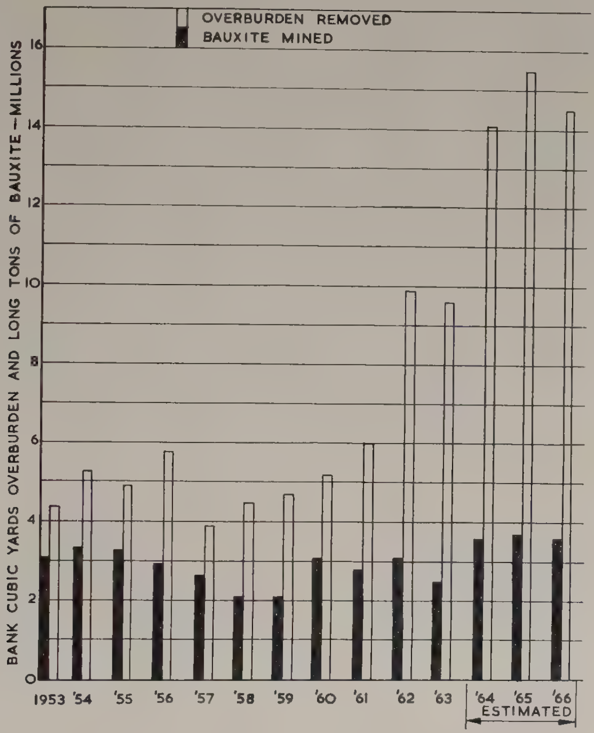


Fig. 1.—Mining and stripping quantities for the period 1953–1966.

STRIPPING METHODS

Hydraulicking

The first wet stripping method employed at Demba was hydraulicking. It consists of breaking the overburden from the face by the impact of water from a relatively low-pressure monitor jet, and allowing the sand

and clay thus released to flow freely into the disposal area. It is feasible only when the disposal area is at a lower elevation than the stripping face and at a relatively short distance from it. The most efficient runoff gradient in Demba's mines is approximately 5 per cent, but lower gradients can be attained if the sand–water ratio is decreased, or some sort of hydraulic elevator added and the discharge channelled through a pipe or sluice. The present applicability of this method to Demba's operations is restricted; most of the overburden amenable to hydraulicking has already been removed by this means. Hydraulicking is a very simple operation, the equipment required consisting only of a water pump, a pipeline and monitors. The pumps used by Demba are electrically driven 250-ft-head centrifugals, normally taking water from the Demerara River, which is not too far distant from most of the mines. In the most recent hydraulicking operation the slurry was channelled to run into the mined-out area where the sand settled out, the clarified water returning by gravity to the main storage basin to be pumped back to the monitors.

Eventually, as the distance from the stripping face to the disposal area increased, the effective runoff gradient from the hydraulicking face was maintained by the use of an ejector, which acted as a hydraulic elevator and delivered to a discharge line. The ejector, a venturi with an opening permitting solids to be drawn into the jet, was designed and fabricated locally. It was adequate when short discharge lines were used. The operation is simple; the overburden can either be pushed over the ejector by bulldozer or washed into it by monitor. The water is delivered to the ejector at approximately 70 lb/in^2 pressure and the water pumps used are similar to those employed in normal hydraulicking, although the pressure required for the ejector is higher than required to wash sand out of the stripping face. Hydraulic ejecting is only practical in sandy overburden which contains no lumpy material.

Ordinary hydraulicking, as one would suspect, is the most economical method of overburden removal Demba has employed. The costs were about 5½d. per bank yd^3, but rose as high as 10d. when an ejector was used. These costs are for 1950 and have not been adjusted to allow for inflation.

Hydraulicking and Sand Pumping

As previously mentioned, hydraulicking combined with sand pumping (Fig. 2) has been used since 1952 and has proved to be a successful means of removing sandy overburden, but it does add to the difficulties of Demba's type of mining operation. It is best completed, therefore, before mining starts in an area, but this is seldom practicable. The water pumps are normally located on the river bank, although in some cases paddocks have been built in the mining areas and the clarified

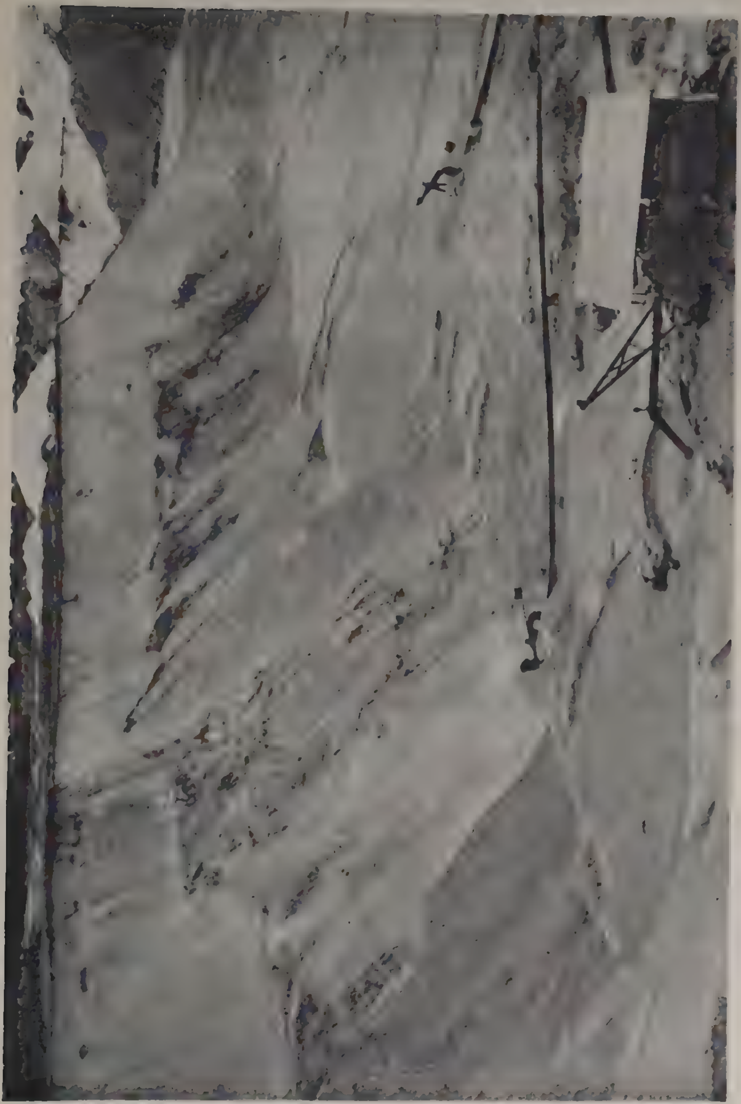


Fig. 2.—Hydraulicking and sand pumping. Monitor on left is washing sand back to dredge pump, monitor on right is washing back to second dredge pump (not in picture). Per cent solids in dredge pump discharge is controlled by raising or lowering suction.

discharge water impounded and pumped to the monitors from them. Two monitors of 3-in nozzle diameter are generally used with each dredge pump; these are set up in the face, and the monitor water, after breaking down the face, runs as a slurry to a sump previously excavated in clay. A dredge pump, driven by a 500-hp diesel engine, is located on the edge of the sump and pumps the slurry to the disposal area at the rate of about 8000 U.S. gal/min. Water lines are normally 24 in. in diameter and discharge lines 18 in. The pressure at the monitors is approximately 40 lb/in^2—sufficient for the removal of normal sandy overburden. The efficiency of the low-pressure monitor jets drops rapidly when clay beds are encountered in the overburden. Each sand pumping unit, besides the raw water and slurry discharge pipelines, consists of two 300-hp water pumps, two monitors, and a dredge pump (usually a 12-in pump with a 14-in suction and 12-in discharge which is rapidly increased to 18 in.). Under favourable conditions such a unit is capable of removing overburden at the rate of 1 200 000 bank yd^3 per year.

The combination of hydraulicking and sand pumping is a very attractive method of stripping the overburden in excess of the thickness the draglines can side-cast—provided the sand to be hydraulicked is unconsolidated and does not contain clay bands. Another prerequisite, of course, is available space for impounding and settling the sand water slurry. Under these conditions, when discharging through a level pipeline not exceeding 4000 ft in length or its equivalent, the combination of hydraulicking with sand pumping can excavate and dispose of sand overburden at a cost, including depreciation, of 10d. to 1s. per bank yd^3.

From an investment viewpoint sand pumping is particularly attractive. Demba has just decided to purchase a bucket-wheel excavator system of 2 000 000 bank yd^3 capacity, including 6000 ft of conveyor, for an estimated installed cost of £600 000. Hydraulicking and dredge pump units, including pipe, to perform the same work are estimated at less than £300 000, installed. The bucket-wheel excavator was chosen because the overburden was not ideal for hydraulicking, suitable disposal areas were not available, and the bauxite to be uncovered, lying as it does about 50 ft below drainage level, will present a mine drainage pumping problem difficult enough even though a dry stripping method is being used.

Dragline Stripping

Dragline side-casting of overburden into the mined-out area is quite a normal stripping method (Figs. 3 and 4). Railroad mining and tropical rainfall, however, impose conditions not normally associated with an operation of this type. Because the mining shovels load directly from the ore face into railroad cars, close to its base, the dragline bucket must swing over the railroad track and not block it with spillage; also, adequate space for drainage trenches must be maintained between the toe of the spoil pile and the bottom of the bauxite bed. In addition, a ledge must be left between the stripping face and the mining face in order to provide a buffer against any overburden breaking away from the stripping face and falling into the mining area. The net effect of having to provide for the ledge, railroad track and drainage is a considerable reduction in the effective reach of the dragline boom. Many modern mining operations, faced with Mackenzie stripping conditions, would haul by lorries to a central railroad loading yard, but crude bauxite becomes stickier each time it is handled. Transporting the bauxite by lorry would certainly make for more effective utilization of the dragline reach.

The overburden removed by the dragline is usually the clayey portion which, as spoil, has an angle of repose of 30–35°. The base of the spoil piles, however, is saturated and presents little resistance to shear or plastic flow; in addition, the tropical downpours erode the new piles at a rapid rate. Underlying the spoil piles is a soft kaolinitic clay which also shears readily and consequently is not competent to carry the load of the dragline spoil dumps. When this kaolinitic clay fails under load, the spoil piles slump. In order to reduce the incidence of floor clay subsidences, and the resultant spoil pile slumps, a sand base for the spoil piles is laid down by the dragline before the clay stripping begins.

The draglines operate best from a sand base and where the wheel excavator has cut into the clay horizon the area is backfilled with about a foot of sand. Operating on the slippery clay can be dangerous because the tub of the dragline tends to spin out of control on completion of a swing and the machine itself may slide, with possibly disastrous results.

A disadvantage of dragline stripping is that side-casting limits the possible stripped ore reserves, required to provide for an unexpected dragline breakdown, to less than the length of the mining face—in practice about half this length, depending upon the dragline's position in the mining and stripping cycle. A minor disadvantage is that the dragline buckets are very rough tools with which to finally uncover a flat-lying orebody. Therefore the last stripping phase is a bulldozer clean-up of the top of the bauxite.

Four large draglines are now in operation. The first machine, which began stripping in 1952, is diesel-powered and has a 200-ft boom. The second machine is a similar type, but carries a 220-ft aluminium boom. Each of these machines carries a 9-yd^3 bucket. The aluminium boom has given good service; there has been one failure of the steel boom. In 1960 a diesel-powered machine, carrying a 235-ft steel boom and a 10-yd^3 bucket, was commissioned. The latest dragline stripper to be added is electrically driven and has a 195-ft steel boom and a 15-yd^3

bucket. The capacities of the draglines in use vary from 2 200 000 to 3 750 000 bank yd^3 per year each. The two oldest draglines have done a considerable amount of walking; the stripping in five mines, located over a 6-mile stretch, was done by a single dragline. The second dragline commissioned has made three walks, each of 30 miles. Walking the machines long distances, according to the manufacturers, has no detrimental effect so long as the terrain traversed is reasonable and the machine is operated properly. The oldest dragline has had an accident and will soon require extensive repairs to the main frame; the second machine is now showing signs of failure in the same main frame beams. If the manufacturer's statement on the effect of walking is correct, this defect would appear to indicate a weakness in the design at this point.

Dragline stripping costs are presented here and refer mainly to the cost of side-casting clay into the mined-out area, and therefore are not directly comparable with wheel-excavator or dredge pump costs. The average cost since 1952 has been 9d./bank yd^3. The cost, including depreciation calculated on a 10-year basis, amounts to 1s./bank yd^3.

Bucket-wheel Excavator Stripping

When it was decided to use a bucket-wheel excavator for stripping (Figs. 3–8), Demba had to choose between the stacker-type bucket wheel and the type of bucket wheel normally used in conjunction with a conveying system.

The stacker-type wheel was eliminated because circumstances dictated that the wheel excavator's lowest operating level would have to be 60 ft above the top of bauxite, and stacking the material to be removed, on top of the dragline spoil piles, would have required an extremely long and expensive stacking boom. A further complication was the fact that, as previously mentioned, the clays underlying the existing spoil piles, being relatively unstable, are liable to fail when loaded. Moreover, stacking in the mined-out area would have tied the wheel-stripping advance to the mining advance. For these reasons, stacking did not seem feasible under the operating conditions which would have prevailed and consequently a bucket-wheel excavator and conveying system was chosen in preference.

The stripping plan for the first wheel commissioned required the use of a dragline for the lowest 60 ft of overburden—which is usually mainly of clay—and of wheel excavator and conveying system for the overburden above the dragline working level—which normally consists of white fine- to coarse-grained sands. The wheel excavator would remove the overburden lying above a working floor 60 ft above the top of bauxite, working parallel to and a few hundred feet ahead of the mining face. The wheel excavator dump would be beyond the mining limits.

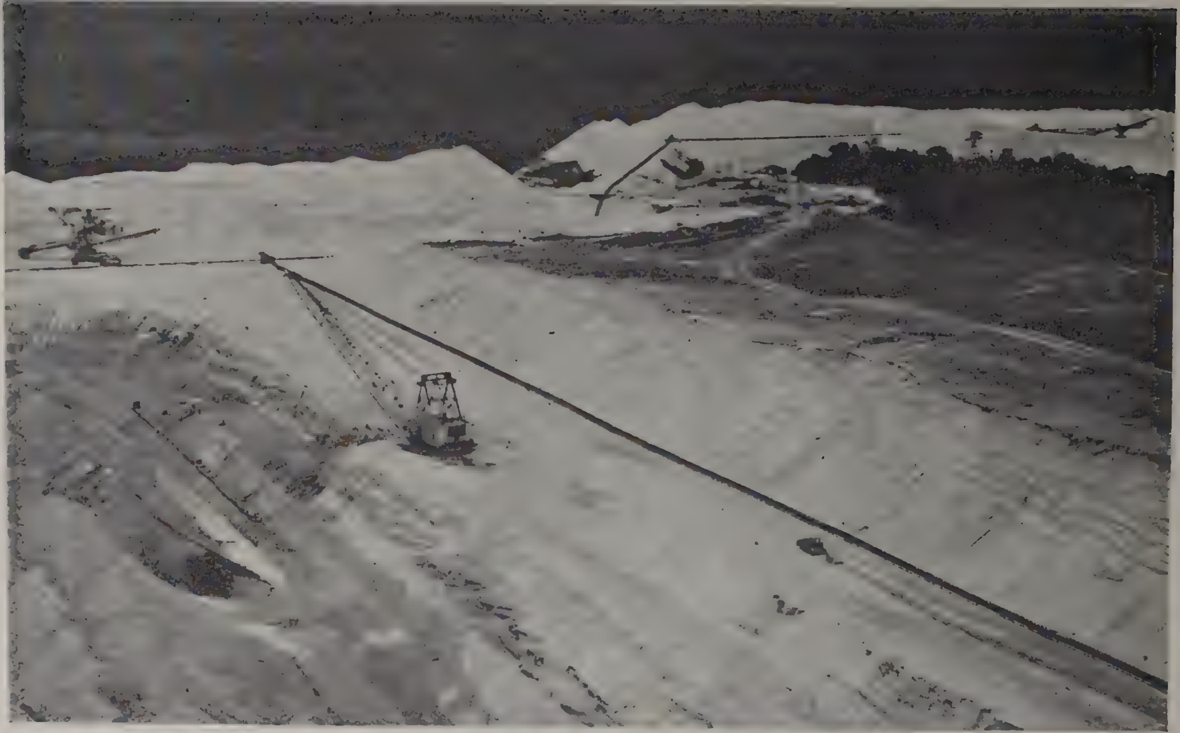


Fig. 3.—Arrowcane mine. Bauxite train being loaded at mining face and dragline side-casting into mined-out area left centre. Bucket-wheel excavator positioned to discharge to no. 2 conveyor, upper left, while no. 1, the face conveyor, is being moved and levelled at the stripping face which extends from the upper left to right foreground. Portions of no. 3 and no. 4 conveyors are visible in upper right-hand half of picture. Conveyor no. 5 and the mobile slewing conveyor can also be seen at the extreme upper right-hand of the picture.

In this arrangement the conveying system consists essentially of four metre-width belt-conveyor units, each capable of operating at a speed of 3·5 m/s. These components are: a face conveyor (no. 1), 3000 ft long; a connecting conveyor (no. 2), 1500 ft long; a rising conveyor (no. 3), 1500 ft long; and a dump conveyor (no. 4), also 1500 ft long. The layout of these units is shown in the photograph of the Arrowcane mine (Fig. 3). The face conveyor, or no. 1 conveyor, extends (from the upper left centre to the right foreground of the picture) along the stripping face. At right angles to it, and at the same elevation, is the connecting conveyor, or no. 2 conveyor, visible in the upper left. The connecting conveyor, which is shortened as the stripping face advances, delivers the spoil from the face conveyor to the rising conveyor, or no. 3 conveyor. The rising conveyor, seen in part in the upper right centre of the photograph, lifts the spoil up to the level of the dump conveyor, or no. 4 conveyor, which extends across the upper right background of the photograph. The dump conveyor is swung, pivoting about the tail pulley, so that the spoil dump takes the form of a segment of a circle, or fan, as the stripping operation proceeds. The positioning of the no. 3 and no. 4 conveyors is determined by the relative positions of the stripping face and overburden disposal area. Auxiliary equipment to increase the disposal range of the dump conveyor consists of a mobile double-wing slewing conveyor with a span of 50 m (Fig. 7), and a skid-mounted sand thrower (Fig. 8). The mobile slewing conveyor can also be seen in the upper right background of Fig. 3, near the end of the dump conveyor. This unit is used, on occasion, to increase the range of the bucket-wheel excavator at the face, at which times it is replaced on the dump by a sand thrower (Fig. 8). Conveyor no. 5 is also visible in the same part of the photograph (Fig. 3), positioned between conveyor no. 4 and the mobile slewing conveyor. It is about 100 ft long, installed at right angles to no. 4 conveyor from which it carries the spoil to the mobile slewing conveyor. It is required so that operations can continue during periods of subsidence on the dump.

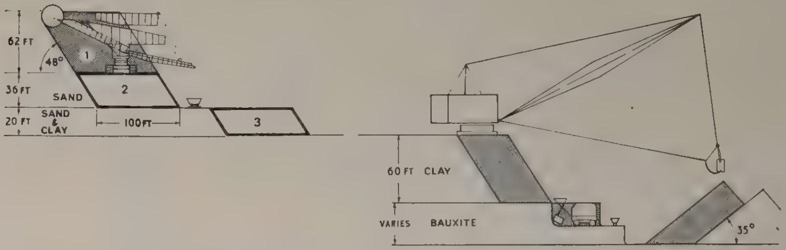


Fig. 4.—Typical stripping section wheel and dragline.

The subsidences are slow, occurring, on the recently deposited ground, in movements of a few inches at a time, and no. 5 conveyor crosses the subsiding dump at right angles to the 'strike' of the ground movements. The conveyor alignment is therefore little affected by these movements, which would force continual shutdowns of no. 4 conveyor if occurring under it. When the dump becomes stable beside no. 4 conveyor, and throughout its length, no. 4 can be pivoted and re-aligned in its new position on stable ground. As no. 5 conveyor is a recent addition, and probably a temporary conveyor, it has not been included in the main components of the system as described earlier in this section.

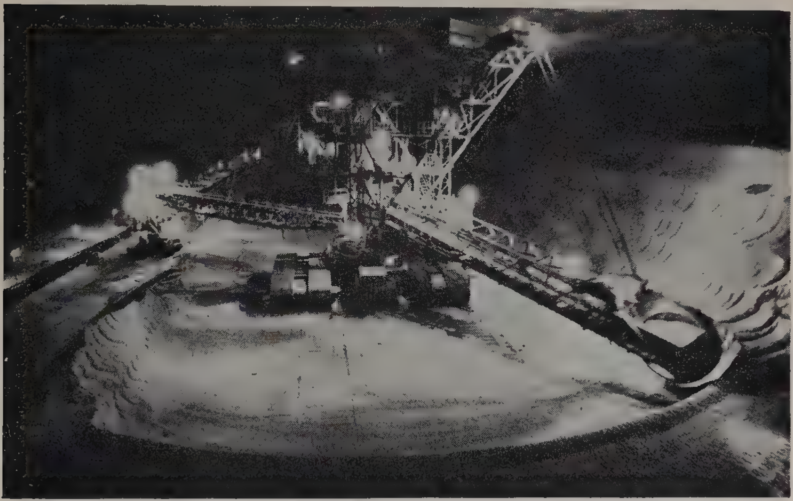


Fig. 5.—Night view of bucket-wheel excavator.

The bucket-wheel excavator (Figs. 5 and 6) carries seven 250-litre buckets and is capable of digging, over a horizontal range of 100 ft, to a height of 67 ft measured from the base of its tracks. As shown in schematic cross-section in Fig. 4, a three-bench stripping cycle is followed. On its first pass the excavator cuts an upper bench in the top of the stripping face, some 60 to 70 ft in height. A second bench, from 26 to 36 ft in height, is cut below the first on the next pass of the machine. These vertical dimensions depend on the surface contours and are so arranged that the floor of the second bench is at the level of the face conveyor.

In the photograph of the Arrowcane mine (Fig. 3) a stripping cycle has just been completed and preparations are under way for the next. It

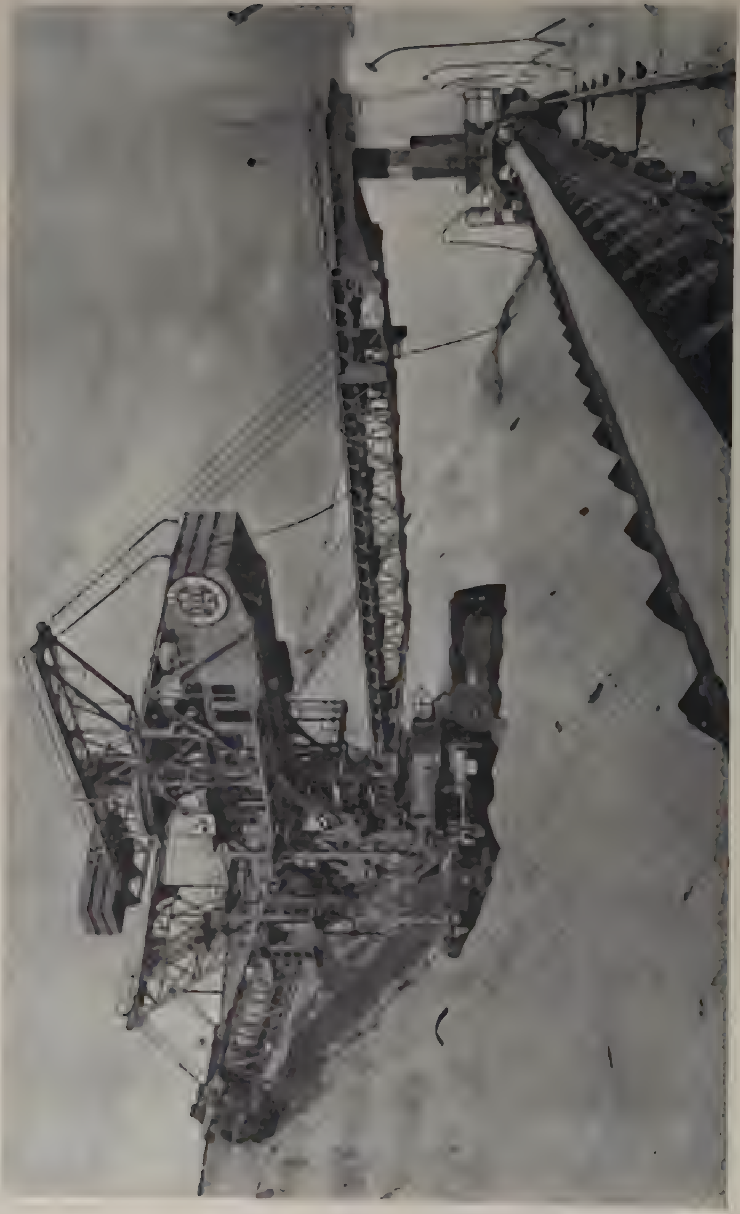


Fig. 6.—Bucket-wheel excavator on bench no. 1.

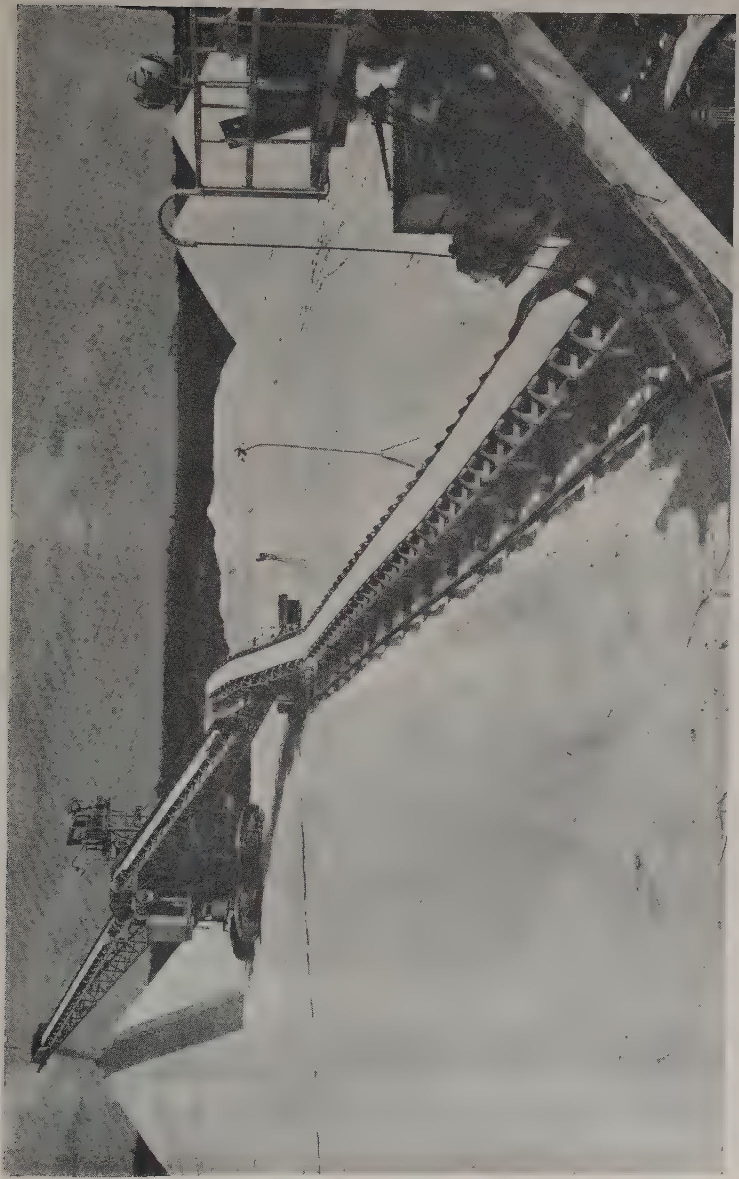


Fig. 7.—Conveyor no. 5 feeding mobile slewing conveyor.

will be noted that the surface elevation increases from left to right along the stripping face, which extends from the left centre to the right foreground of the picture. As a consequence, the floor of the first bench of the new cycle will be started near the head pulley of the face conveyor, in the left centre of the photograph, at an elevation of 26 ft above it. The ramp to elevate the excavator to this level runs beside, and parallel to, no. 2 conveyor. The floor will be continued at this level until the height of the first bench, as it progresses towards the tail pulley, which is out of view in the right foreground of the photograph, reaches 60–70 ft. From here the floor will be gradually raised until it is 36 ft above the level of the face conveyor and then maintained at this elevation for the remainder of the cut. While the bucket-wheel excavator is operating at this higher level, the sandy spoil from its discharge boom will be dropped about 20 ft on to the sloping backplate of the belt hopper of the face conveyor (Fig. 6) in order to give it a forward impetus in the direction of conveyor travel and thus reduce belt wear and improve loading of the no. 1 conveyor. Upon completion of the first bench, the machine will be walked back to its starting position to commence the excavation of the second bench, the floor of which will be kept at the level of the face conveyor. The face conveyor will then be moved, across the 100 ft width of the second bench, to the stripping face and re-aligned and levelled. When the face conveyor is ready to resume operations, the excavator will return to the face-conveyor head-pulley area to dig the

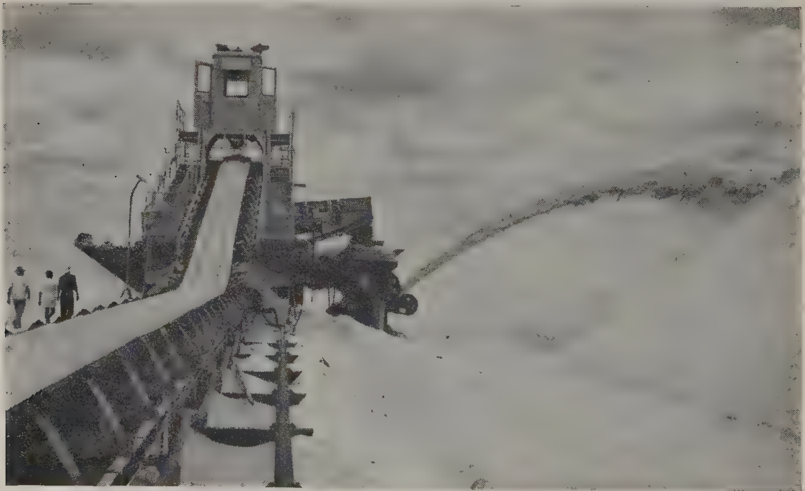


Fig. 8.—The mobile slewing conveyor on the dump is replaced by a skid-mounted thrower during periods when the mobile unit is out of service or required elsewhere (such as for use with the wheel for lengthening the face belt cut). The dump belt tripper discharges to a shuttle conveyor feeding the thrower.

third bench, which is on the opposite side of the face conveyor and has its floor elevation 20 ft below conveyor level (Fig. 4).

Face conveyor moves can be completed in about 2 days, although no serious effort has been made to reduce this time because the bucket-wheel excavator can load to the no. 2 conveyor while the no. 1 conveyor is being moved. The pivoting of the dump, or no. 4 conveyor, also requires about two days but this time, too, could probably be improved. Advantage is taken of the time provided by these conveyor moves to carry out repairs and maintenance to the conveyor systems, belt vulcanizing, power-line repairs or moves, etc. To date, the overall operating availability of the bucket-wheel excavator has been about 60 per cent. From present operating experience the capacity of this machine and its conveyors is estimated to be 4 000 000 bank yd^3 per year and the electrical load of the system is about 900 kW.

Bucket-wheel stripping and conveying costs over the two-year period during which the first bucket-wheel excavator has been in operation have amounted to 9d./bank yd^3, or 1/3d. with depreciation calculated on a 10-year basis.

As previously mentioned, Demba has decided to purchase an additional bucket-wheel excavator system, consisting of a 150-litre bucket-wheel excavator, two mobile conveyors, and 6000 ft of conveyor which will carry a 36-in belt. The duty required of this wheel excavator is 2 000 000 bank yd^3 per year, and it is to work in parallel with an electrically-driven walking dragline with a 195-ft boom and a 15-yd^3 bucket which will side-cast about 40 ft of overburden into the mined-out area. In about three years' time a 70-litre bucket-wheel excavator, with a capacity of 1 000 000 bank yd^3 per year, will be added and will discharge to the conveyor system of the 150-litre bucket-wheel excavator mentioned above. This unit will be available from one of the other mines.

FUTURE STRIPPING

Experience to date with the above stripping methods indicates that future changes will probably be in the direction of reducing, with bucket-wheel excavators, the thickness of overburden to be handled by draglines in order to improve the mining conditions, or else resorting to lorry rather than railroad haulage from the mine face, in which case the draglines could handle thicker overburden. The biggest problems in sand pumping are connected with the occurrence of clay in the sand overburden. Low-pressure water, adequate for sand, is not very effective on the clays; the use of high-pressure water is unattractive from a cost viewpoint. This year a 70-litre bucket-wheel excavator will be used to excavate clay and discharge it to a dredge pump; the wheel simply replaces the monitor. It is possible that in some cases it might be more economical to strip with a wheel excavator discharging to a dredge pump than it is either by hydraulicking to a dredge pump or by the conventional wheel excavator conveyor scheme. This probably will depend on the percentage of solids it is possible to maintain in the pump.

CONCLUSION

Demba is engaged in the highly competitive alumina business as well as the equally competitive abrasive, refractory and metallurgical bauxite businesses, and the conditions under which feed for the alumina and bauxite plants is produced are more onerous than for any of its competitors. The Company has maintained its competitive position, despite the tremendous increase in stripping requirements, by heavy capital investments in the most modern large-scale earthmoving equipment. As a result of these heavy investments stripping costs have been kept down in spite of increasing overburden, rising costs for labour, materials, supplies, etc. Like coal, bauxite is a relatively low-value mineral and, again like coal, is abundantly available in many countries. But for the cumulative mining experience gained by Demba over nearly 50 years of operation in British Guiana, the financial resources at its command and the willingness to invest them in the advanced mining equipment now available, the deeply buried bauxite currently being mined might, like many known coal deposits, otherwise only have remained an interesting mineralogical occurrence instead of being a valuable natural resource.

Acknowledgement.—The author would like to express appreciation to various members of the operating staff of Demerara Bauxite Co., Ltd., in particular to Mr. E. Anderson, chief geologist, for assisting the author in preparing a preliminary draft of the paper, and to Mr. P. A. Snijders, chief mining engineer. He would also like to thank the officers of Demerara Bauxite Co., Ltd., for permission to submit the paper for publication and of Aluminium Laboratories, Ltd., Montreal, for advice on its preparation.

622.271.3:622.349.52(485)

Paper 33

Ranstad—A New Swedish Opencast Mine

G. OLSSON

Mining Engineer, AB Atomenergi, Ranstadsverket, Skultorp, Sweden

---

SYNOPSIS

Sweden is at present erecting a plant for production of 140 metric tons of U_3O_8 per year. The plant will go into production in the middle of 1965. The raw material is an alum shale containing approximately 300 g/t uranium. The shale will be recovered in an opencast mine.

The paper first gives a brief description of the whole plant and the geological conditions, and then the mining methods and machinery used for development and operation are presented.

The mine will have a capacity of 800 000 metric tons uranium shale per year. The bench height of the uranium shale is 11·5 ft, while the overburden height averages about 40 ft; thus the overburden ratio is approximately 3·5:1.

Stripping is done by a Bucyrus-Erie 480-W dragline—the biggest machine of its kind in Scandinavia. Some operation cost figures are presented.

For drilling in rock overburden and uranium shale a new Atlas Copco electro-hydraulic rotary drilling machine is used. Drilling costs are given.

One electric excavator will load the blasted uranium shale into 20–30-ton diesel trucks for hauling to an underground crushing plant.

Sweden's domestic production of uranium is at present restricted to the treatment of a low-grade shale (approximately 300 g of uranium per metric ton). A plant for the production of uranium concentrate is now being constructed at Ranstad, near the town of Skövde in southern Sweden, from raw material recovered by opencast mining. The plant, which will go into production in the middle of 1965, will have a capacity of 140 metric tons of U_3O_8 per year, in an 84 per cent concentrate. The total cost of the plant has been calculated to be about £10 000 000 at the 1961 price level.

GENERAL GEOLOGICAL CONDITIONS

The uranium shale is part of the Cambrian–Silurian stratified rock series of the Billingen mountain. This series consists mainly of sandstone, alum shale, limestone and claystone. The uranium is confined to a shale bed 11·5 ft thick. The bed, which is virtually horizontal, has a very large extension, the total content of uranium in the district being considerable—about 1 000 000 metric tons of uranium.

Normally the zone of high uranium is located deep below the surface, where it can be recovered only by underground mining, but in the vicinity of the Ranstad plant there is a relatively large area of shale with shallow overburden, suitable for opencast mining.

A fact of technical importance is the occurrence of limestone lenses in the uranium shale, which has a heat value of about 1500 kcal/kg.

BRIEF DESCRIPTION OF THE PLANT PROCESS[1]

The uranium shale is hauled from the pit to an underground primary crusher, where it is reduced to 14 in. maximum size. The crushed material is taken by a conveyor belt to a screening plant, located on the top of a storage silo. The material is divided by screening into three fractions, $-\frac{1}{4}$ in., $\frac{1}{4}$–2 in. and 2–14 in., the first being removed to waste and the two remaining fractions entering a heavy media separation plant having two parallel circuits. The limestone inclusions of the shale are removed and the float product, i.e. the dressed uranium shale, is then crushed by impact-type crushers to $-\frac{1}{8}$ in., the particle size being controlled by screens in the crusher circuits. The product is then ready for chemical treatment. This begins with an oxidation period in the open air, after which the shale is charged into large concrete tanks lined with acid-resisting bricks where the material is leached with sulphuric acid; the uranium is dissolved, together with several other elements, and the solution is passed through a selective ion-exchange column, which absorbs the uranium. After a series of eluating and extracting processes the uranium is recovered in the form of a dry concentrate, containing 84 per cent U_3O_8. The leaching residue is hauled by trucks to a special disposal area. The total waste quantity is approximately the same as the quantity of incoming raw material from the mine—800 000 metric tons per year. Fig. 1 shows the layout of the plant.

OPENCAST OR UNDERGROUND MINING

The uranium shale in the vicinity of the plant, to be won either by underground or opencast mining, has been investigated by diamond drilling and seismic methods, and a trial tunnelling programme has also been carried out. The result of a comparative calculation indicated that it is more economic to use opencast mining for at least the first fifteen years of production. Since the uranium shale with shallow overburden has a restricted extension, however, it was decided to place the primary crusher underground on a level to suit the future underground mining.

SELECTING THE MOST SUITABLE OPENCAST MINING AREA

In selecting the best mining area, large enough for the first fifteen-year period of production, of the many factors influencing such a

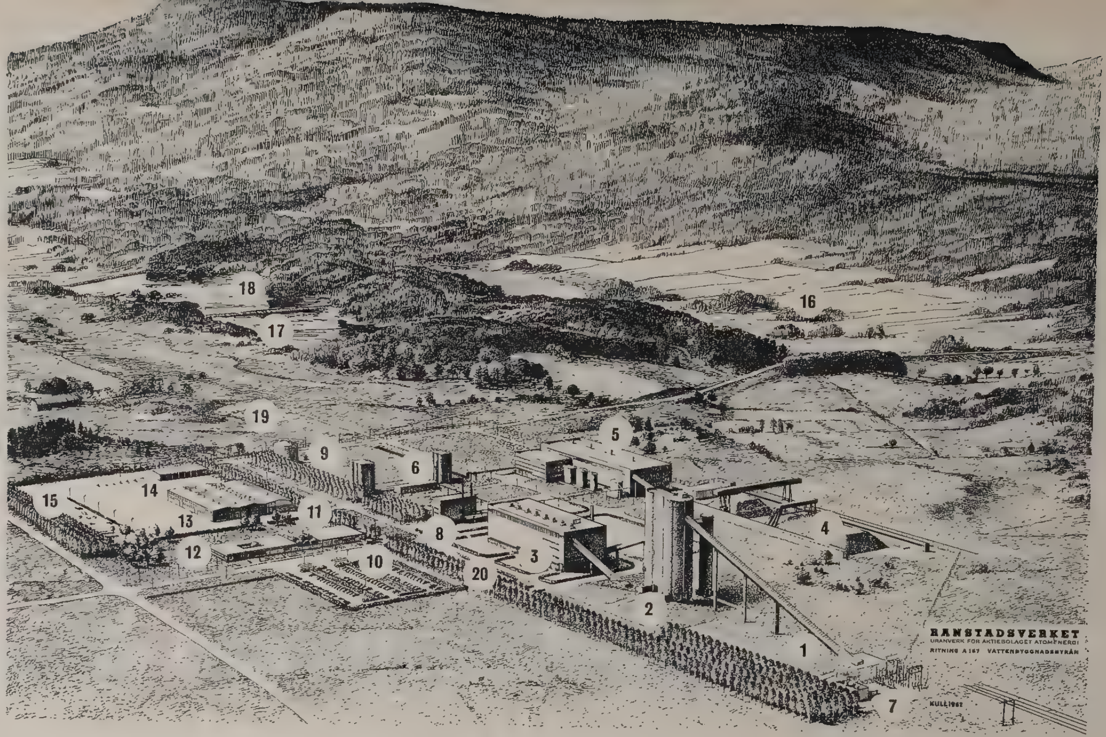

18
17
16
19
5
9
6
15
14
11
13
4
8
12
10
3
20
2
1
7
RANSTADSVERKET
URANVERK FÖR AKTIEBOLAGET ATOMENERGI
RITNING A 157 VATTENBYGGNADSBYRÅN
KULL 1962

Fig. 1.—The Ranstad plant layout.

1. Conveyor. 2. Bunkers and screens. 3. Dressing plant. 4. Weathering pile. 5. Leaching plant. 6. Neutralization plant. 7. Transformer station. 8. Steam boiler and water supply. 9. Oil storage. 10. Main entrance and parking place. 11. Dining rooms, dressing rooms, etc. 12. Office and laboratory. 13. Workshop and storage. 14. Garage. 15. Storage area for materials. 16. Waste area. 17. Water reservoir. 18. Retain basin. 19. Sanitary effluent treatment plant. 20. Tunnel to underground crushing station.

decision—e.g. haulage distance, existing buildings, roads, power lines, etc.—the main factor in this case could be said to be the quantity and distribution of the overburden. The results from both drilling and seismic investigations were used for producing maps showing the depth of earth, rock and total earth-plus-rock overburden, within the whole potential opencast mining area. These maps were then transformed into a 'cost map' to show the relative stripping costs in the area. The calculated costs for removing different types of overburden are required in addition to the overburden depth maps, and at Ranstad the rock overburden has to be drilled and blasted before it can be removed, making the costs for rock stripping three to four times higher than those for earth removing. The result is that the total stripping cost varies considerably depending on the ratio of rock-to-earth overburden depth.

When preparing the 'cost map' a numerical calculation of total stripping costs for each 300-ft by 300-ft unit area was first made, the charts thus formed then being transferred to the 'cost map', which presents the conditions in a more surveyable way. Maps of this type are, in fact, very suitable for evaluating stripping conditions rapidly within a particular area. The accuracy of such a map is, of course, influenced by the difficulty of making an exact estimate of the specifie stripping costs and overburden depth.

The selected mining area covers 435 acres, large enough to meet the requirement of fifteen years of production at a capacity of 800 000 metric tons of uranium shale per year. Fig. 2 shows the average depth of the different types of overburden in the mining area, the stripping ratio being about 3·5:1. The border of the selected area coincides in part with the outcrop of uranium shale (Fig. 3).

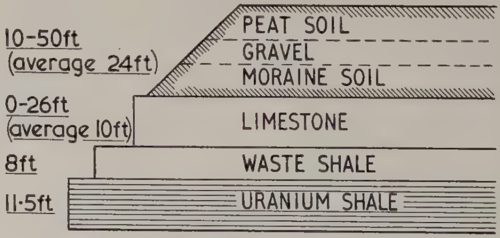


Fig. 2.—Geological section in the first opencast mine area; overburden ratio approximately 3·5:1.

DESCRIPTION OF THE PLANNED MINING SYSTEM

The mining method is based on using one walking dragline for all stripping, and mining the uranium shale by benching at normally only one working place, a system similar to those common to several

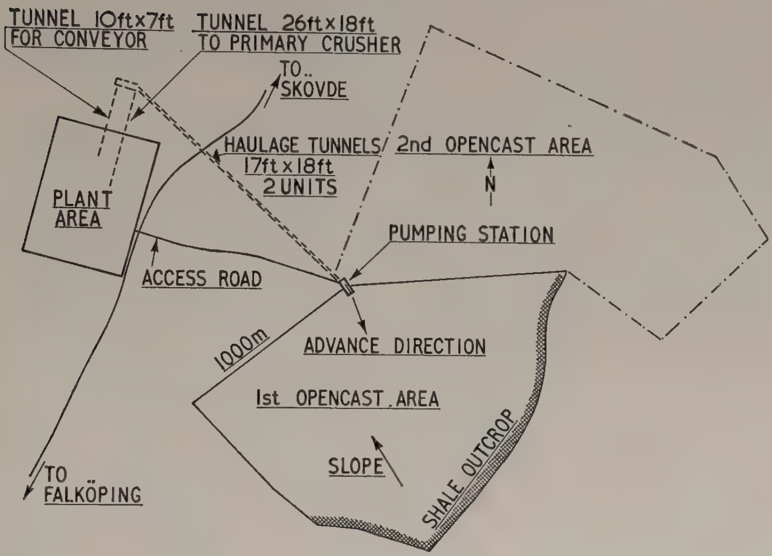


Fig. 3.—Location of plant and mine areas.

British opencast mines, including iron mines in Northampton and Lincolnshire.

The mine is opened up alongside the NW border, i.e. closest to the mill (Fig. 3), the reason for starting at this side of the deposit being mainly that all future workings will be to the rise, which facilitates the drainage of the site. This is a fact of great importance, since the area is marshy, and it has been possible to arrange a permanent drainage station near the tunnel entrance during the development stage. Another advantage is that the costs for haulage roads are cut to a minimum during the same period. On the other hand, however, the quantities of overburden to be handled in the initial gullet are bigger than would have been the case if starting along the outcrop side of the area.

As the mining front advances, the central haulage road on the bottom level will be gradually extended through the area.

Fig. 4 shows the principle of the mining method. The stripping is going on at one side of the central road at the same time as the benching of uranium shale is carried out on the other side. When the fronts reach the borders of the area the machines return to the central road, where a new cut is started. If the dragline should reach the road, after having performed a cut, before the benching on the other side is finished, the dragline must wait, because haulage on the bottom would otherwise be stopped. If, on the other hand, the benching advances faster than the stripping, the benching may continue on the same side

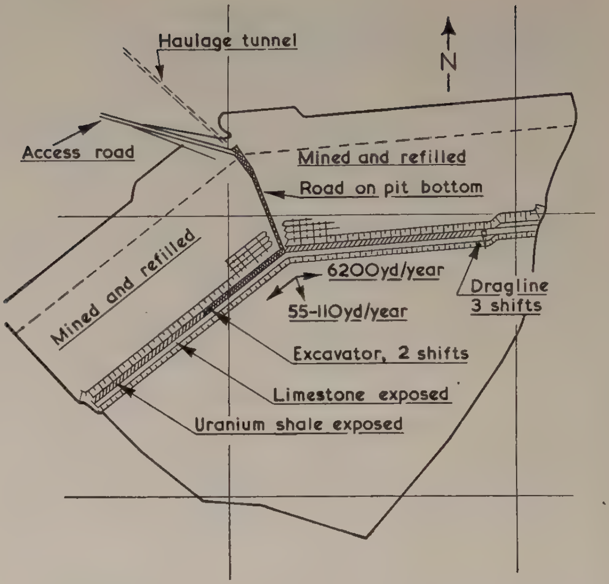


Fig. 4.—Plan showing mining method.

of the road as the stripping without any difficulty. All stripping will be done by the dragline only, without rehandling, and the uranium shale will be hauled by diesel trucks direct from the loading spot to the underground primary crusher.

Overburden Removal

For removal of the overburden an electric walking dragline will be used with a working weight 660 tons, jib length 195 ft, jib angle 40° and medium-type buckets of 14-yd^3 and 16-yd^3 capacity. During full production this machine will be run three shifts per day for five days a week. The dragline will have to handle the following quantities per year:

	Average during the 15-year period	*Average during approx. 2 months*
Earth overburden, yd^3	1 200 000	1 300 000
Rock overburden, limestone and waste shale, yd^3	880 000	1 200 000
Total yd^3	2 080 000	2 500 000

The machine has a dumping radius of 165 ft, which allows a cut width of 65 ft minimum when the machine is used in the way shown in Fig. 5—65 ft having been considered the cut width necessary for truck hauling on the bottom. The dragline was purchased in 1961 and has been used for development work since January 1962. Before deciding on type and size of machine for stripping, thorough calculations were made, such as investment costs, operating costs, practical capacities when handling different types of overburden, the possible benefit of a larger dumping radius than the minimum required. These investigations showed, briefly, that the best choice would be a walking dragline with a capacity sufficient to meet the requirements represented by the short time/average volume figures, and a dumping radius which would guarantee the minimum cut width with no rehandling, or only for very short periods. One fact that complicates the choice of walking draglines is that the gaps between available sizes are very wide, the next machine in size to that chosen being about double the weight and price.

When using a single machine for stripping it is essential that the machine should have a reasonable reserve capacity and that the stripping system used should make it possible to keep a good distance between stripping and mining. The dragline chosen for the Ranstad work has a fairly good reserve capacity in normal conditions. In addition, the mining system will give about two months' lead for the dragline during the early years of production, and although this lead will decrease during the last years of the 15-year period, owing to the shape of the mining area, the overburden depth will decrease at the same time, so giving the machine a better reserve capacity.

As shown in Fig. 5, the rock overburden must be drilled and blasted before being handled by the dragline. The blasting will be performed by vertical or slightly inclined holes and the blasted rock will be directed towards the spoil side. The dragline will move on the pile of blasted rock, the rock pile previously having been levelled by a bulldozer and if necessary covered with earth material without rock pebbles. The required jib length will obviously be much reduced when the rock pile is used as the operating road for the dragline. This method is practised on several British sites, but with the difference that the blasting in the British mines is not great and leaves the rock surface in almost the same condition as before blasting. The rock overburden at the Ranstad site being considerably harder than the limestone in most of the big British strip mines, it is necessary to blast with fairly strong charges in closely spaced holes to reduce the rock to pieces suitable for effective handling by the dragline. This method of blasting also has the effect of throwing some of the rock directly into the spoil area without dragline handling. This introduces the possibility of avoiding dragline work within certain limits as, when blasting solely to obtain the necessary reduction, about 15 per cent of the rock does not need further handling by the dragline.[2]

Fig. 5 also shows the distribution of rock and earth material in the

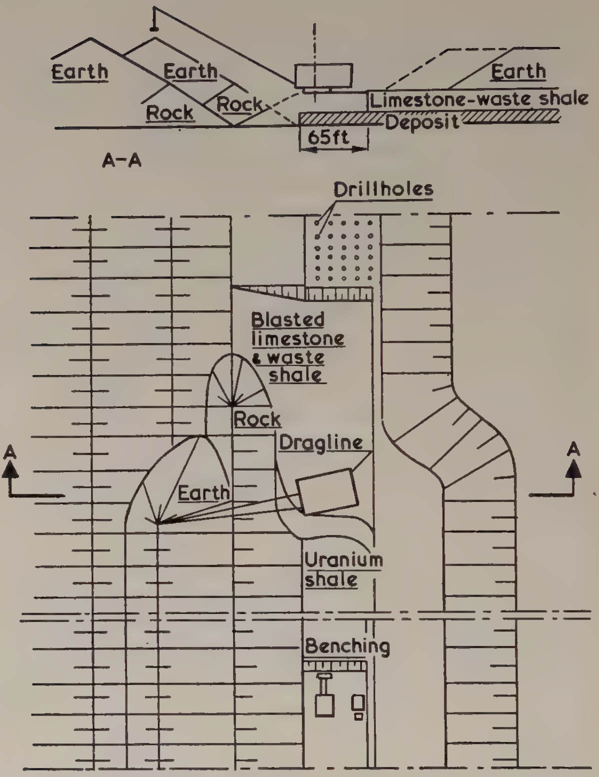


Fig. 5.—Plan detail of stripping method.

spoil heap. By dumping the rock in advance of the earth, the rock is at the bottom and the earth at the top of the spoil, with the two advantages that the rock will be readily pervious to water, thus facilitating drainage of the pit, and the earth on the top will be comparatively easy to restore into forest or farming land.

For blasting the rock overburden 80-mm holes will be drilled with 10-ft by 10-ft spacing and depth varying between 9 and 38 ft, averaging about 20 ft. Two electro-hydraulic rotary drills at present employed on development work will be used. The holes will be charged with ammonium nitrate–fuel oil (AN–FO) explosives, which will probably be prepared at the site.

Benching of Uranium Shale

The benching will involve drilling, blasting and loading by excavator into trucks. The same type of drill will be used as for overburden drilling. This machine will have a very considerable reserve capacity and can be used as a standby unit for the overburden drilling machines.

The excavator, electric-powered, will have a bucket capacity of about 5 yd^3. The trucks will probably be of the normal rear-dump type with a capacity of 20–30 metric tons. The machinery for the benching has not yet been purchased.

Haulage Roads

The hauling distance from excavator to primary crusher will vary between about 1500 and 3000 yd. The slope is towards the crusher (about 4 : 1000). From the excavator to the central road the trucks will travel on a temporary road, on the mine bottom, prepared from plant waste products spread and levelled by bulldozer, with a life of about 2 months at most. The central road, including the tunnel, will on the other hand have a long life and so will be of first class quality with asphalt surface, constructed for heavy traffic at high speed. The tunnel consists of two parallel units, one for loaded and one for empty trucks, each unit 17·3 ft wide and approximately 18 ft high, which is sufficient for up to 40-ton rear-dump trucks. Sodium vapour discharge lights will be used for illuminating the tunnel. At the tunnel entrance a proper concentration of lights will be arranged to avoid dazzling the truck drivers' eyes.

Power Supply

Electric power will be used for all machinery except trucks and

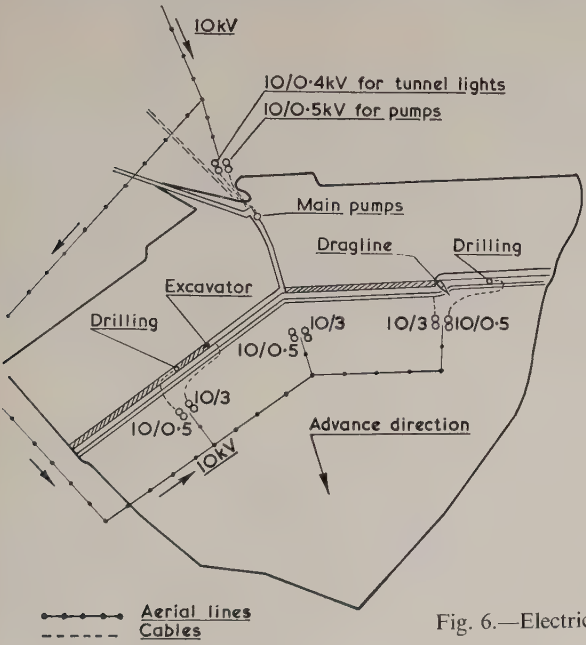


Fig. 6.—Electric power supply system.

tractors, etc. The power supply system has been designed to be flexible and not to disturb the mining operations. The machines are fed by cables from three movable transformer stations located on the advance side, as shown in Fig. 6. The aerial lines feeding the stations can be disconnected easily and moved when mining has advanced far enough. The dragline and the excavator receive 3 kV, and other machinery 500 V. The incoming voltage to the movable transformer stations is 10 kV. By this power system layout cables are avoided on the spoil side and the cable length is reduced to a maximum of about 450 yd for each machine.

Drainage

The permanent main pumping station for drainage has been established at the tunnel entrance where the lowest point of the uranium shale bed is located. Most of the surface and ground water in the mine will flow along the cuts and through the rock spoil to this station without pumping, except at local hollows. If the rock spoil proves to be insufficiently permeable, pipelines for drainage can be arranged on the pit bottom to prevent roads and working places being flooded. At present the pumping station is equipped with three submersible pumps, automatically controlled by float switches, each pump having a nominal capacity of about 5 yd^3/min at 82-ft lifting height. One of the pumps is sufficient for the incoming water under normal conditions.

DEVELOPMENT OF THE OPENCAST MINE

Planning

Preparation work at the site was started late in 1960, when an access road was constructed from the plant area to a point near the future tunnel entrance. Digging and blasting for the slope down through alluvials, limestone and shale to the bottom level of the tunnel were completed in May 1961, after which the tunnel work could be started. Up to that point the work had been carried out by contractors, but the development has been continued by AB Atomenergi personnel and equipment. Apart from the tunnel the main development work—the excavation of the initial gullet or box-cut—has been going on since January 1962.

The box-cut is located at the lowest side of the uranium shale bed to facilitate drainage, as already mentioned. The principle used in establishing the box-cut is shown in Figs. 7 and 8. The width of the exposed uranium shale in the cut is 82 ft. In addition to this, a constant width of 65 ft of limestone surface must be exposed on the advance side and 40–80 ft on the spoil side, depending on the total overburden depth. Thus the total width of exposed limestone will be between 187 and 227 ft.

It can be seen that all the overburden from the initial gullet is

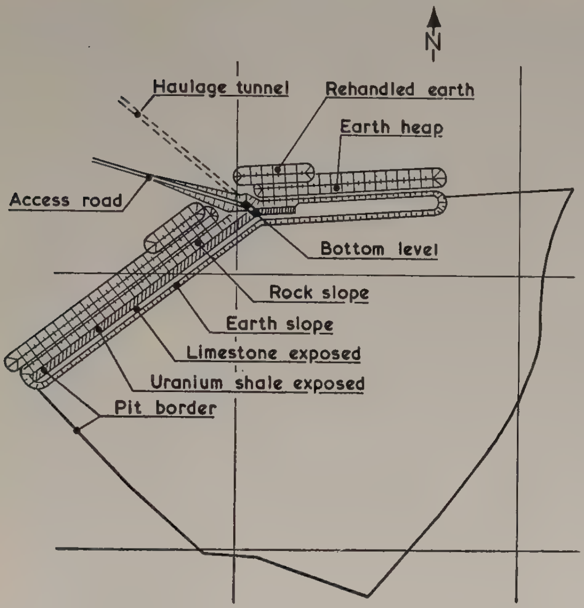


Fig. 7.—Plan drawing of box-cut excavation during the development period.

heaped outside the mining area, which means that the uranium shale along the border is deeply buried. Future opencast mining of this area would be very expensive and probably will never take place, but this loss has been considered to be of minor importance in comparison with the extra cost during the first years of production which would be incurred if the overburden from the box-cut had been spoiled on the advance side for later rehandling several times over.

The total length of the box-cut will be 2290 yd. How much of this length it will be necessary to excavate during the development will depend on the benching speed during the first period of production. The ideal would be to complete the box-cut at the same time as the

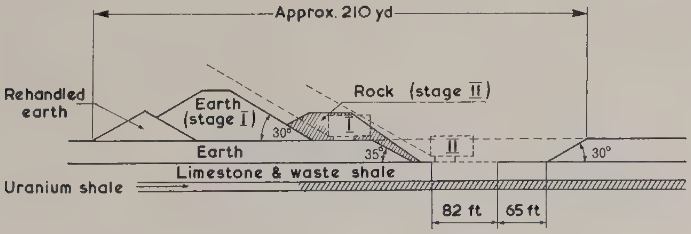


Fig. 8.—Section of box-cut excavation.

first benching cut was finished, which would mean that stripping could continue without interruption and that the lead for the stripping was kept at the maximum. Because the actual time for completing the box-cut during simultaneous benching is difficult to estimate, the length of the box-cut during development should include a sufficient margin.

In the present plan for the development period the box-cut will be entirely completed on the SW side of the central road (1090 yd). On the NE side of the road the limestone surface will be exposed to a length of 760 yd and the uranium shale surface will be exposed to a length of 120 yd. Modifications will be made in the light of experience during excavation of the box-cut. Overburden volumes to be handled during development have been estimated as:

Earth overburden	1 270 000 yd^3 (bank measure)	Rehandling (earth)	290 000
Rock overburden	270 000		
		Total	1 830 000 yd^3

The average depth of earth overburden in the box-cut is 24 ft, and of rock overburden, 25 ft. The excavation of the box-cut is carried out on the basis of one 9-hr shift per day for 5 days a week.

Equipment and Personnel

Equipment comprises one walking dragline, Bucyrus-Erie 480-W Special, 16-yd^3 bucket, 195-ft jib, electrical; one crawler tractor, International TD-25; two rotary drilling machines, Atlas Copco BVB 101 (one standby unit).

On the staff are one mining engineer, who is also in charge of tunnelling, and one foreman. There is a total of seven labourers, four on the dragline and on drilling and blasting, one on the crawler tractor, and two for miscellaneous work.

Details of the Dragline Job

As shown in Fig. 8, the excavation of the box-cut is carried out in two steps, the earth overburden is removed first, then the rock overburden. During earth excavating, the machine walks on the original ground level but during rock excavating it walks on the pile of blasted rock. Rehandling has to be done when the earth overburden depth exceeds 28 ft, depending on the reach of the dragline. In addition, it is necessary to rehandle the topsoil of peat character, if the depth of this material exceeds 3 ft. The peat is not hard enough to support the dragline when walking and digging, nor can it stand the pressure from overlying heavy alluvials which could cause sliding of the spoil into the mine; therefore the peat must be moved to the outside of the spoil heap. This rehandling is comparatively cheap because the peat is very easy to dig.

Fig. 9 shows the dragline during earth excavation in the box-cut.

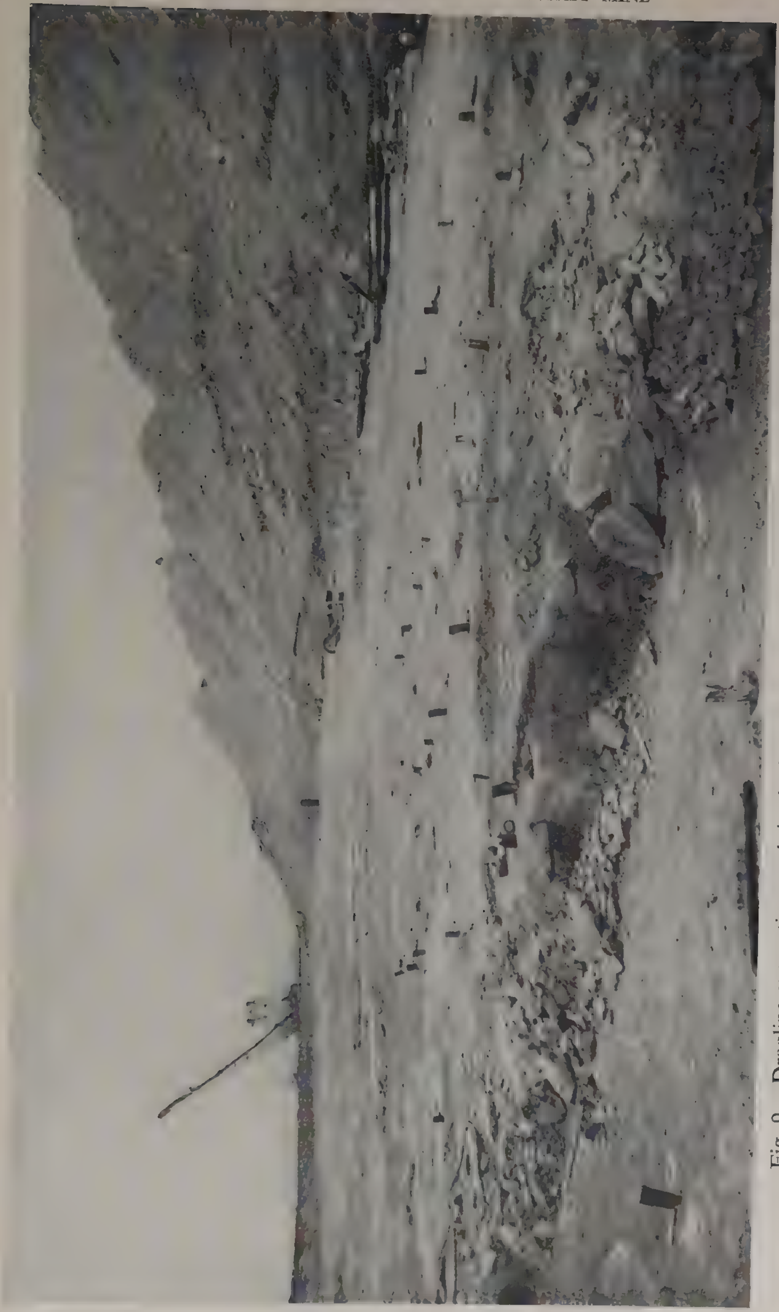


Fig. 9.—Dragline excavating earth in the box-cut (in the foreground plugged drill-holes in the limestone).

The practical capacity of the machine has so far been about 450 yd^3 bank measure per hour when excavating alluvials, using 14- and 16-yd^3 buckets alternately. The bucket factor is about 0·93 and the operating efficiency 0·73. When conditions are favourable the capacity can be increased considerably, 790 yd^3 bank measure per hour having been reached during a special capacity test. Its capacity for extended work on rock excavation is not yet known, but during a test the result was 490 yd^3 bank measure per hour.

There are two crews of two men identically trained for the dragline and they are also in charge of the drilling and blasting, changing over weekly from excavating to drilling and vice versa. The labourers are paid at an hourly rate of about 7 Sw. crowns per hour (1963). The operating costs for the dragline during the excavating of 960 000 yd^3 of earth, and rehandling 180 000 yd^3 of earth and 28 000 yd^3 of rock (all in bank measure) have been 270 000 Sw. crowns, not including administration, management and social costs. The operation cost per yd^3 bank measure has thus been on an average 0·23 Sw. crowns. The dragline operating costs are distributed as follows:

	Per cent		Per cent
Wages for the crew	20·9	Bucket teeth	8·6
Wages and materials for repairs	15·7	Lubricants	2·2
Spare parts	11·4	Electric power	29·2
Ropes	12·0		
			100·0

The crawler tractor works in co-operation with the dragline and for this machine the operating costs during the first 2000 hours are 35 Sw. crowns per hour. One new set of tracks has been purchased during the period.

Details of Drilling and Blasting

Fig. 10 shows the Atlas Copco BVB 101 rotary drilling machine. Its main features are:

Weight	Approx. 13 tons
Mounting	Crawlers
Power	500-V electric power, total about 100 kW
Hydraulic systems	One closed-circuit system for rotation, feeding and traction; one open-circuit system for jacks and mechanized rod-coupling device

Feed	Double chains, feed length 20 ft, thrust up to about 4 tons
Rotation speed	From 0 to 250 rev/min
Levelling	4 hydraulic jacks
Bit diameter	Up to approx. 4 in.
Rod diameter	2½ in.
Rod length	20 ft
Flushing	Compressed air
Crew	One man

When drilling in the box-cut the average hole depth is about 27 ft, which means that two rod lengths must be used. During production,

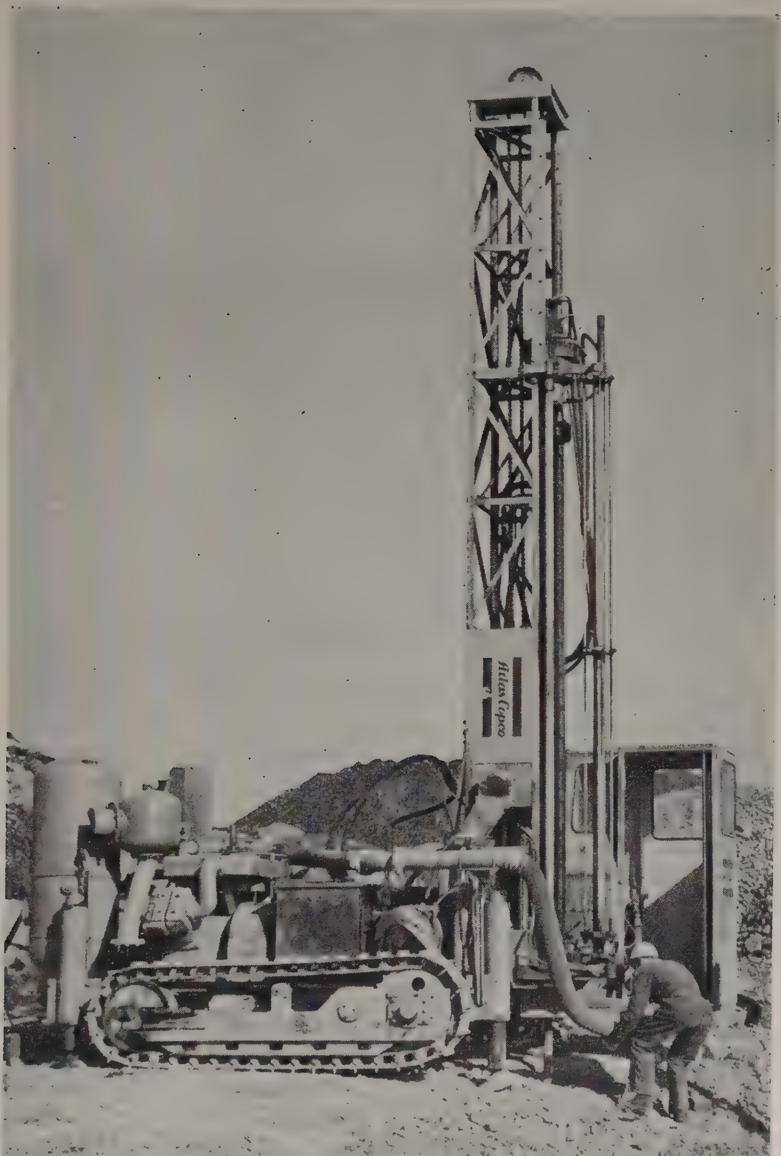


Fig. 10.—Atlas Copco BVB 101 electro-hydraulic rotary drilling machine.

however, about 50 per cent of all holes can be drilled with only one rod length, giving a considerable increase in capacity.

All holes are drilled vertically, with spacing of 8 ft by 10 ft without staggering. Blasting conditions would be more favourable if the holes could be inclined, but the stratified rock has a tendency to bend and destroy the drill rods if drilling is not done vertically.

The drilling speed is about 3·5–5·0 ft/min on an average for the box-cut drilling down to the uranium shale bed. Occasionally, the speed can increase up to 6·5 ft/min.

A bit of 3-point tungsten carbide tip is used, the bit life at present being about 1000 ft (1963). The main problem concerning bit life is that the limestone and shale contain a small amount of pyrite crystals, which wears the bits considerably.

The operating cost for drilling 185 000 yd^3 of limestone and shale is 130 000 Sw. crowns, not including administration, management and social costs. The corresponding total hole length is about 75 000 ft. Thus the drilling costs for this period are 0·72 Sw. crowns per yd^3 bank measure or 1·73 Sw. crowns per ft of hole. These costs, which refer to the first period of drilling, are distributed as follows:

	Per cent		Per cent
Wages for operator	21·8	Bits and rods	39·0
Wages and materials for repairs	13·8	Lubricants	4·3
Spare parts	6·1	Electric power	15·0
			100·0

The cost of a certain amount of trial drilling is included, as is the cost of purchasing a suitable number of bits and rods for storing.

As blasting in the box-cut has to be done without mucking, the consumption of explosives is higher than for blasting during the production period as described above.

The holes are charged with a slurry-type explosive called Reolite at the bottom, and above this AN–FO in plastic bags is used as column charge. The bottom charge has a length of about 10 ft, while the column charge ends about 7 ft below the surface. Drilling dust is used for stemming. The charge is ignited with a detonating cord through the hole, down to the slurry column. The total weight of explosives per hole is about 77 lb on average. Electric millisecond caps are used for igniting the detonating cord. The consumption of explosives is about 1·05 lb/yd^3 of rock bank measure.

REFERENCES

1. HORMANDER, O. and GELIN, R. How Sweden is developing its low grade U_3O_8. *World Min.*, **16,** March 1963, 28–31.

2. Casting overburden with explosives. *Coal Age*, **66,** March 1961, 78–82.

622.343(831.3):622.271.3

Paper 34

Open-pit Mining at Chuquicamata

R. C. BECKER, B.S. (Min. Eng.)
Assistant Vice-President and Assistant General Manager, Santiago, Chile
E. W. WITCOMB, B.S. (Met. Eng.), M.I.M.M.
Assistant Manager, Chuquicamata

SYNOPSIS

The early history and development of the mine are given briefly, together with current production figures. Sections on labour, water and power then follow.

The orebody and present mining practice are described in detail, and future mining plans are outlined in general terms.

Chuquicamata mine lies in the province of Antofagasta in the Republic of Chile, at latitude 22 03 South and longitude 68 55 West of Greenwich. The mine is in the Atacama Desert, in the foothills of the Andes Mountains, at an elevation of 9200 feet above sea level (Fig. 1). The climate is extremely dry, although temperatures are moderate, and sunshine prevails almost throughout the year. Strong and continuous winds are a feature of the desert winter between March and September.

The copper deposit of Chuquicamata has been known for a long time. Legend tells that the first Spanish 'Conquistadores' passing through the region in 1535 used copper from Chuquicamata to shoe their horses. Development of the mine began in 1912, but before then many small mines worked the deposit, which produced high-grade hand-sorted ore. Old maps show the towns of Punta de Rieles and Placilla—no longer in existence—and thus indicate the early activity.

In 1910 A. C. Burrage, a Boston banker, believing that the ore was amenable to treatment by the Bradley Process, obtained options on mining claims and on other rights in the area. Unable to continue financing the operation himself, all assets were grouped under the ownership of Chile Exploration Company. Chile Copper Company formed in 1913, owned all the assets of Chile Exploration Company. The ownership of Chile Copper Company was divided between Burrage and the Guggenheim Brothers. In 1914 additional financing by means of a $15 000 000 bond issue was obtained to bring the mine into production for an ore treatment rate of 10 000 t/day and a possible copper production of 100 000 000 lb of copper per year. Copper was first produced by Chile Exploration Company in 1915.

In 1923 Anaconda Company purchased Chile Copper Company by

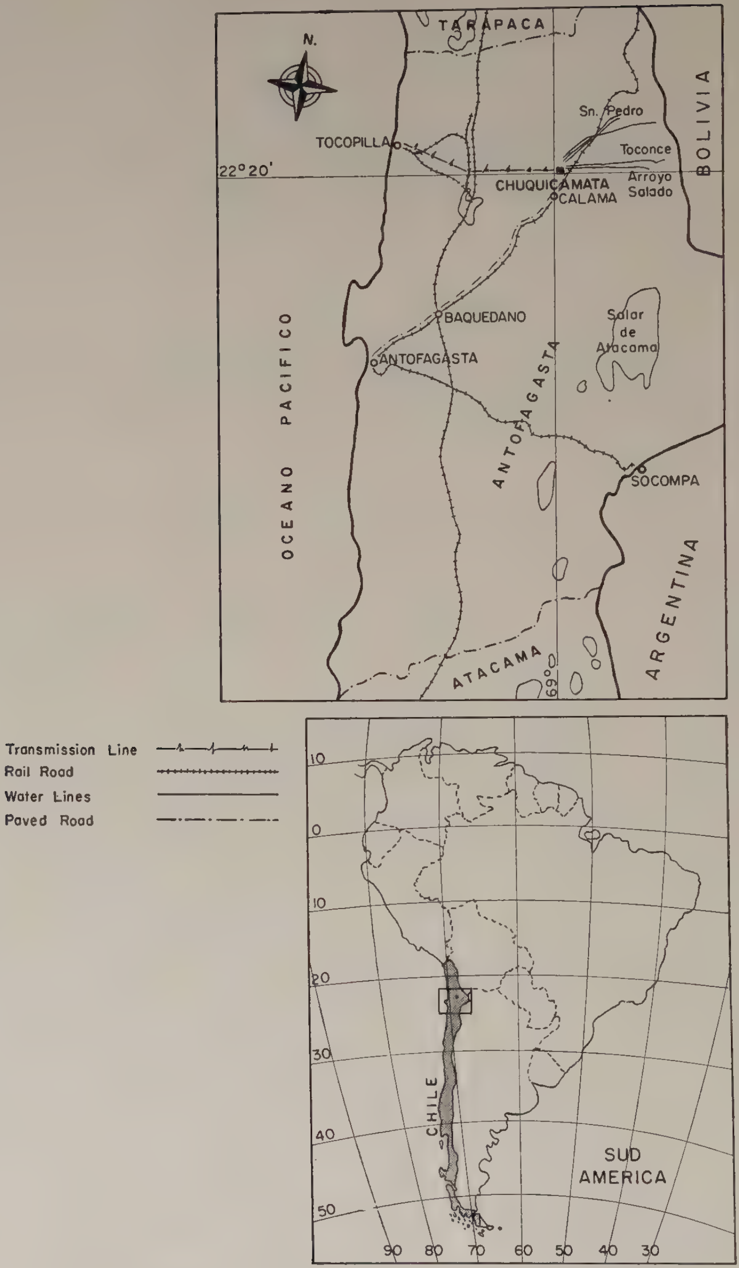


Fig. 1.—Sketch-map showing location of Chuquicamata mine.

buying the shares of the Company held by the Guggenheim Brothers. Anaconda Company now owns 99·43 per cent of the stock of Chile Copper Company. In 1925 a heavy capital investment was made in order to modernize and enlarge the operation, and the project increased copper production to 450 000 000 lb of copper per year. During the Second World War a further expansion took place to raise production to 500 000 000 lb of copper per year.

In 1948 a fourth construction project was undertaken—the provision of metallurgical facilities for the treatment of the sulphide ore encountered in the mine. These facilities have been continuously enlarged to take up the production requirements, as copper production from the treatment of oxide ores by leaching decreases. Substantial investments have been made in order to modernize the operation and to provide the means for mining in the pit from greater depths.

At the present time the Chuquicamata operation is an integrated one in which, for the first seven months of 1964, a daily average of 175 546 t of material between ore and waste was removed in the mine. During this seven-month period 178 405 t of copper have been produced —70 470 t from the electrolysis of solutions obtained from leaching oxide ore and 107 935 t from the smelting of sulphide copper flotation concentrates. Of the production obtained from the smelting of concentrates, 49 823 t have been refined in the electrolytic tank house, making a total electrolytic production of 120 293 t for the first seven months and 58 112 t of blister. During the same period 1 434 000 lb of molybdenum were produced in the form of concentrates of molybdenite—the sulphide of molybdenum. The electrolytic copper is cast to wirebar and is sold on the world market under the trade mark CCC, the largest portion going to Western Europe. Blister that cannot be refined at Chuquicamata is sent to the U.S.A. for refining.

All the Company's mining and metallurgical operations are carried out in Chuquicamata, which is connected by a British-owned railroad with the port of Antofagasta 263 rail kilometres distant. Power for the operation is generated by a steam turbine plant at the coastal town of Tocopilla, the transmission lines being 138 kilometres long.

The mining operation at Chuquicamata is centred in one single mineralized area, currently producing over 600 000 000 lb of copper per year.

LABOUR

In June 1964 6730 unionized people were employed directly by Chile Exploration Company. They were distributed between the three localities as follows: Chuquicamata 6369; Tocopilla 288; and Antofagasta 73.

In addition, there is a supervisory group of 371 persons. Supervisory

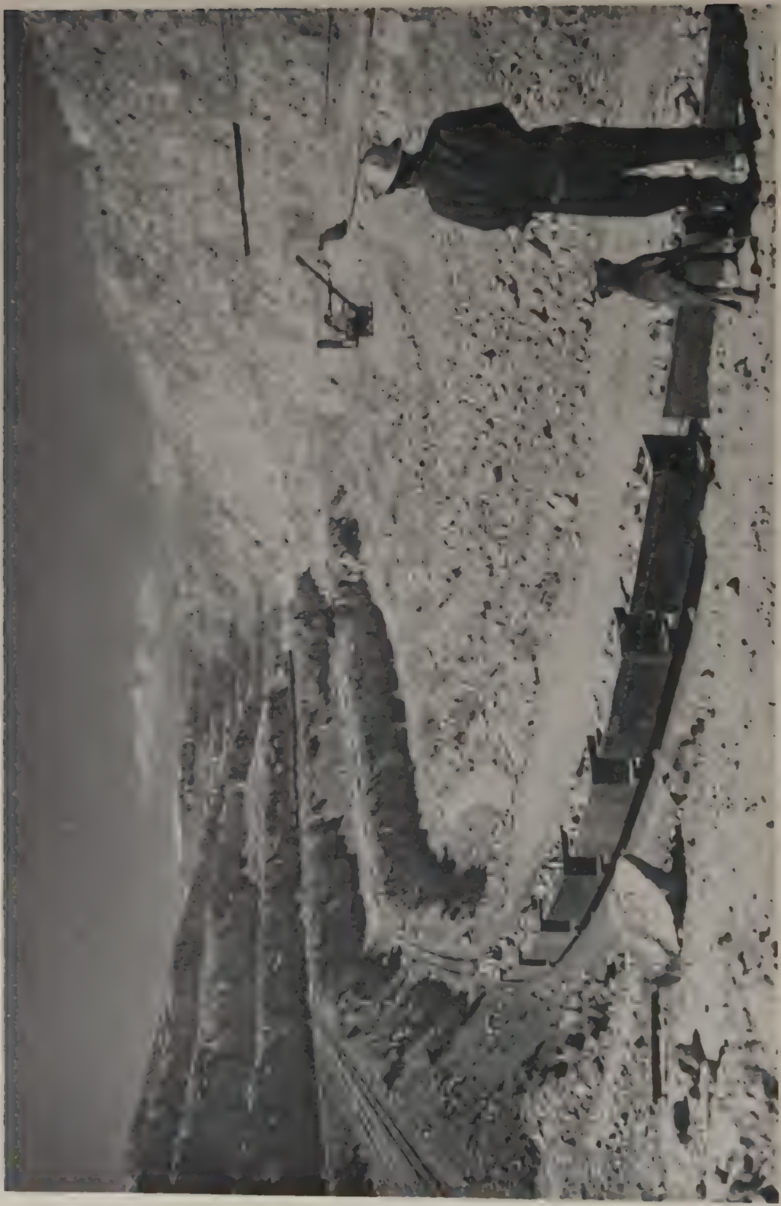


Fig. 3.—View of mine from South.

personnel, as distinct from the rest of the employees and labourers of the Company, are not unionized. Of the supervisory personnel, 273 are Chilean by birth or naturalization, 52 are citizens of the U.S.A., 15 are British subjects, and the remainder are citizens of other countries in Europe, South and North America.

In Antofagasta, which is one of the largest cities of Chile, the Company maintains no housing or social amenities for its employees. In Tocopilla the Company maintains a townsite to the south of the town which is adjacent to the power plant and which contains a company store and medical centre.

For the mining operation at Chuquicamata the Company owns and maintains the townsite of Chuquicamata, where the majority of the working force lives. Sixteen kilometres to the south of Chuquicamata is the town of Calama—the administrative centre of one of the three Departments of the Province of Antofagasta. In Calama at the Departmental level are the Government Administrative offices and officials. The town of Calama has a population of some 38 000 inhabitants, and about 1500 of the workers directly employed by the Company live there.

The townsite or mining camp of Chuquicamata has modern houses, churches, a modern hospital, cinemas, shops, sports stadiums, clubs and schools.

By legal definition there are two classes of workers in Chile: the larger group is called 'obreros' (labourers) and the smaller 'empleados' (employees). Truck drivers, engine drivers, shovel operators and most foremen are employees. Tracklayers, mechanics and electricians are labourers. Employees are paid monthly and have certain benefits through Government-controlled Retirement Banks that lead to substantial retirement pensions. Labourers are paid daily wages and their retirement benefits, through Government Social Security, are low compared with those of the employees. Of the 6369 people directly employed by the Company, 1698 are employees and 4671 are labourers.

Table I shows the breakdown of the average labour cost for an employee (a diesel-electric locomotive engineer) and a minimum rate labourer (a platelayer). Some of these costs, such as the base rate, production bonus, night-shift bonus, general bonus, seven-day-week bonus and special Sunday bonus, are the man's direct take-home pay. Family allowance, student allowance and family compensation are only averages and will vary depending on whether the man is married and on the number of dependent children. Housing allowance and transportation bonus are paid only in exceptional cases. Vacation and vacation bonus is the average cost of vacations pro-rated per shift. Other figures are the average cost per shift of the fringe benefits that the worker and his family receive.

TABLE I.—*Average labour cost of employee and labourer*

	Diesel-electric locomotive engineer	Platelayer
Base rate	22.04	10.82
Production bonus	5.50	3.84
General bonuses	0.64	1.41
Night-shift bonus	0.31	0.27
Seven-day-week bonus	—	2.38
Special Sunday bonus	0.02	0.02
Vacations	1.84	1.15
Vacation bonus	0.56	0.55
Christmas bonus	0.56	0.54
Gratification profits participation	3.24	2.20
Family compensation	6.40	5.50
Student allowance	0.85	0.38
Family allowance	—	2.37
Housing allowance	0.19	0.46
Transportation bonus	0.05	0.11
Social laws	10.27	2.70
Welfare benefits (house, light)	5.18	4.98
Indemnity years service	2.51	0.81
Total per shift in Chilean escudos	60.16	40.49
Equivalent in U.S. dollars	25.07	16.87

WATER

To the east of Chuquicamata is the Loa River system, some of its tributaries being saline water and some sweet. The source of the water is a series of springs in the high Cordillera. Industrial water used in Chuquicamata is obtained from two saline tributaries of the Loa River. Drinking water is obtained from two sweet water springs near the frontier with Bolivia. From these four sources the water is conducted to Chuquicamata by five pipelines. The total water brought in is 66 000 m^3 per day, the shortest pipeline being 59 kilometres long and the longest 104 kilometres.

All used domestic water is piped to a sewage plant and the effluent is available for use in the leaching plant and concentrator when necessary. The only water that need be discarded is the final discard solution from the leaching plant and final concentrator tailings thickened to 50 to 52 per cent solids by weight.

Rain and snow are very rare in the Atacama Desert. When rain does come, however, the flash floods that can occur cause severe damage to the pipelines. A crew is maintained at a village on the Loa River, 30 kilometres to the east of the mine, and is available for any emergency.

POWER

The main source of power for the operation of Chuquicamata is the company-owned power plant at Tocopilla. Fuel oil is shipped in from the South of Chile or the Caribbean by tankers and discharged to storage tanks by submarine pipe. Several major changes have been made in the plant since it went into service in 1915.

In 1937 the Tocopilla power plant consisted of 24 boilers and 8 turbo-generators. The installed rated generating capacity was 100 000 kWh. The equipment operated at a pressure of 215 lb/in^2. In 1937 six boilers were replaced by one large one, designed to operate at a pressure of 800 lb/in^2 and providing 30 000 kW of energy in the steam. The steam from the new boiler is sent to a new 'Topping' turbine of 10 000 kW and the exhaust steam is used in the original low-pressure units. In 1959 a 50 000-kW turbo-generator was put into service. The boiler for this new unit is designed for a pressure of 2000 lb/in^2. The turbine operates with a reheat cycle and the generator is hydrogen cooled. Further modernization is planned and will lead to the withdrawal of all the original low-pressure equipment.

A small power plant at Chuquicamata utilizes the steam generated from waste heat boilers installed on the reverberatory furnace at the smelter. Maximum capacity of the waste heat power plant is 21 000 kW.

Apart from supplying the Company's mining and domestic needs, power is sold to the towns of Calama and Tocopilla and to other mining companies in the locality. Power generation at Tocopilla totalled 73 000 000 kWh in July 1964, the power being transmitted to Chuquicamata by three lines. Transmission voltage is 100 000 V, three-phase. The incoming power is received at the main sub-station for distribution. Power to the mine, oxide plant and townsite is distributed at 5000 V to the secondary sub-stations where further transformation is made to 500 and 220 V. Power to the sulphide plant sub-station is distributed at 100 000 V from the main sub-station, where the power is transformed to 13 800 V for distribution to the secondary sub-stations in this plant. At this point there is the link with the power generated at the waste heat power plant.

DESCRIPTION OF THE OREBODY

The Chuquicamata orebody can be described as a porphyry deposit. The host rock is granodiorite porphyry, into the fractures of which was introduced an enrichment of chalcopyrites and pyrites. A fluctuating water-table resulted in partial to total leaching and oxidation and thus the sulphide minerals are generally overlain with oxide minerals such as antlerite, kröhnkite, brochantite and cuprite, antlerite being the most abundant copper mineral in the oxidized zone. This is a hydrated basic copper sulphate readily soluble in dilute acid. Molybdenum in the form of molybdenite occurs in the sulphide zones, and as molybdates in the oxide zones. The occurrence of molybdenum in the orebody is irregular.

The orebody has a nearly oval shape with a sharp cutoff to the west by a pronounced fault, and is limited in other directions by fading of the mineralization. Mineralization occurred close to the surface, which allowed ore to be produced by open-pit mining with only a small amount of stripping of waste.

The present pit is two miles long and is from 660 to 1400 yd in width. Twenty-four benches of varying heights have been developed, the topmost at 9856 ft and the bottommost at 8803 ft elevation above sea level.

To date, 925 000 000 t of material have been mined from Chuquicamata, 500 000 000 t of ore having been treated for its oxide mineral content by leaching and 125 000 000 t of sulphide ore having been treated by flotation. The remaining 300 000 000 t of material have been waste. Over the whole of this period the waste to ore ratio has been 0·48 to 1·0.

Sulphide ore shipments to the plants were started in 1952, after concentrating and smelting facilities were set in operation at a capital expenditure of $125 000 000.

Ore reserves as at February 1957 were:

	Million tons	*% Total copper*
Oxide ores	120	1·32
Mixed ores	25	2·25
Sulphide ores	450	1·63
Total	595	1·59

The estimated ore reserve below the lowest pit level now contemplated, which would be mined by the block-caving method, is 375 000 000 t of sulphide ore at 1·73 per cent total copper.

Copper mineralization in the sulphide ore zones currently being worked is predominately chalcocite with minor quantities of covellite and less chalcopyrite. Mixed ores are those ores which contain sufficient oxide and sulphide mineralization to be economically worth treating by leaching to recover the oxide values and subsequently by flotation for recovery of the sulphide copper values.

PRESENT MINING PRACTICE

Present practice is to develop new benches 43 ft high initially, this being the range of the mechanical shovels used in discharging the loaded material into rail-cars on the bench above. On achieving sufficient width to allow for the introduction of track, shovels with shorter booms replace those of longer range, and with the similar development of a second 43-ft bench the first is eliminated, an eventual bench height of 86 ft thus resulting.

Until 1956 all blast-hole drilling was done with churn drills. With the advent of rotary-type drills, four machines (from three different manufacturers) were purchased. Today, after purchasing four additional machines of the type found most suitable from among the original four, approximately 3600 ft of $9\frac{1}{2}$-in and 12-in holes are being accomplished daily by seven rotary drills. Formerly, 22 churn drills were required to obtain the same footage.

Drill patterns vary with the hardness and structure of the rock to be broken. In opening up new benches where a free face does not exist, holes are drilled at the intersections of an equilateral-triangular pattern $19\frac{1}{2}$ ft equidistant from each other, the whole pattern covering a 30-ft by 275-ft rectangle. Along open faces two rows of holes are drilled at 28 to 33 ft centres; first-row holes are 26 ft from the existing bench toe and second-row holes are staggered 23 ft behind the first-row holes. All holes are drilled 10 ft below the elevation of the bench being developed.

Engineering control is exercised throughout: firstly, in laying out the pattern to be drilled for a shot and staking the location for each hole, along with the depth to which it should be drilled and, secondly, in surveying the tops and toes of the bench face in order to calculate accurately the burden on each hole so that the proper amount of explosive will be charged into it.

Ammonium nitrate prills mixed with fuel oil (AN–FO) is the standard explosive used at Chuquicamata, except where the use of a TNT–AN water-resistant slurry or canned explosives becomes necessary in wet holes.

The Chuquicamata porphyry at depth is without fractures and is capable of holding water. The water comes from its use in the operation of churn drills and from lunchrooms, etc., and flows into the broken rock, covering the bottommost bench, and appears in the blast-holes of a new bench being opened up.

Pre-mixed prills and oil are charged in blast-holes from trucks equipped with a tank, a measuring device, and a compressor to supply air for the discharge of the prills through a flexible hose. Detonating fuse is passed through intermittently spaced primers of 42 per cent strength dynamite or the newer pentathol discs. The detonating fuse from each hole is tied to a trunk line interspaced with 15-millisecond delay mechanisms so arranged that each hole, on detonating, has not one, but two free faces towards which to break.

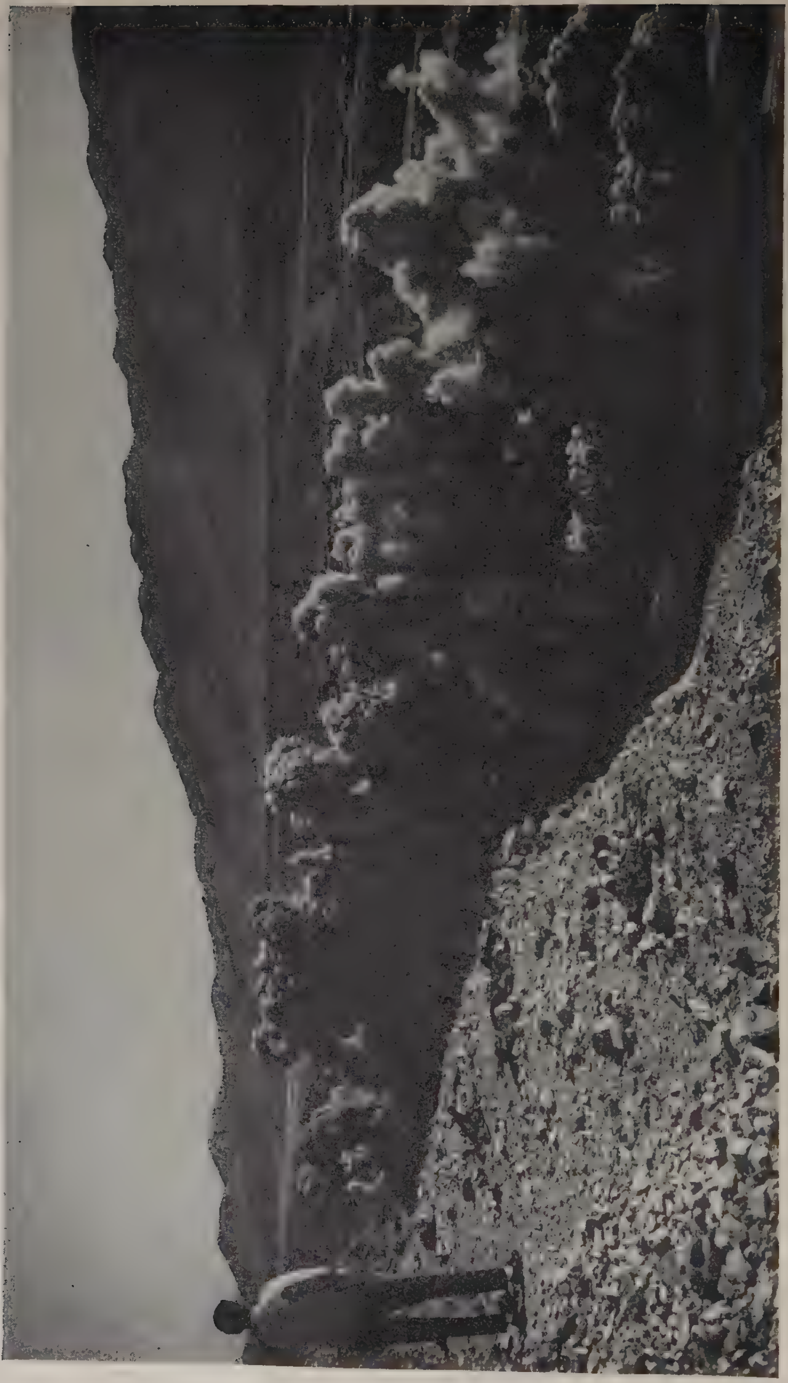


Fig. 4.—Blast-hole shot in waste breaking 1 700 000 t of material: Shot included two benches.

While blasts of more than 1 500 000 t of material have been effected (Fig. 4), the average blast is planned for breaking from 200 000 to 300 000 t of ore or waste. Depending on a number of factors a 250 000-t blast would entail from 60 to 85 blast-holes and, at the very low factor of 2·7 t broken per pound of explosive (which is an indication of the extreme hardness of Chuquicamata rock), would require the use of 46·3 t of explosive. Secondary blasting, always a necessary evil in open-pit mining, has decreased to almost half since the introduction of AN–FO.

Of the 170 000 t of ore and waste loaded daily, 77 per cent is moved by rail; tracklaying and track maintenance are items of major importance to the efficiency and cost of the overall pit operation. With 60 miles of standard-gauge track to maintain, with track having to be removed and relayed before and after each blast and with track having to be moved after each shovel pass is completed, mechanization has been introduced wherever possible.

Bulldozers and road-graders prepare road-beds; prefabricated sections are laid in place by self-powered rail cranes; ballast, consisting of ½-in (to fine) oxide ore tailings, are spread through bottom-dump cars; and levelling and tamping is done with automatic equipment.

The 24 mechanical shovels in operation are electrically powered. Because of the difficult digging conditions they have been de-rated below the capacities assigned them by their manufacturers. Eleven older units, acquired between 1927 and 1949, are equipped with 4½-yd^3 dippers, while eight units brought into service between 1956 and 1959 have 6½-yd^3 dippers. Purchased in 1961 and 1963 are three 8-yd^3 shovels and two long-range development shovels, dating from 1949 and 1963, are equipped with 11- and 8-yd^3 dippers respectively. An average 8-hour loading rate for all shovels is 4150 t, with a high of 5000 t attributable to the newer, larger units and a low of 3250 t to the older, smaller ones.

Shovels are serviced, maintained and repaired by motorized crews of roving mechanics and electricians whose vehicles are equipped with two-way radios. All vehicles used by the foremen are similarly equipped and rapid communication therefore keeps delays to a minimum.

Steam haulage, like steam shovels, was only used in the pit from 1915 to 1927, at which date the first electric locomotives arrived. Even so, the transition was a lengthy one, and the last steam locomotive was not retired until 1948, when another transition period began with the purchase of five 85-t, 600-hp electric locomotives with auxiliary diesel generating units. A year ago the last purely electric locomotives were removed from service and third-rail, along with sub-stations wherein a.c. power was converted to d.c., disappeared. Chuquicamata's present fleet of haulage units is made up of the five locomotives previously mentioned, five 180-t, 1750-hp diesel-electrics and 24 others weighing 124 t and equipped with 1200-hp engines.

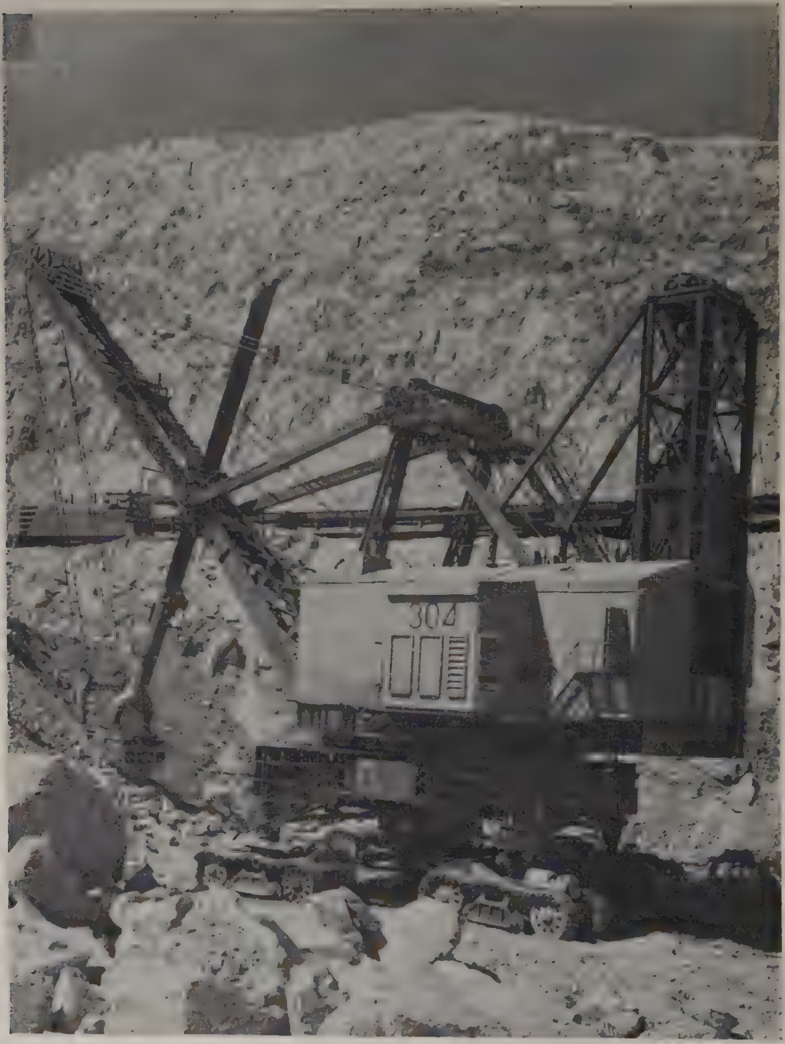


Fig. 5.—Development shovel in lowest bench.

Two types of railroad cars are in use: a gondola-type car which delivers ore to the oxide plant and a side-dump car used for both waste and for delivery of sulphide ore to the concentrator. There are 499 gondola-type cars in service and they have a capacity of approximately 70 t of ore. These cars require a rotary car dumper at the oxide crushers to dump them. There are 296 side-dump cars in service, all equipped with pneumatic-powered dumping mechanisms. Their capacity is 30 and 40 yd^3.

Mine locomotives deliver trains of gondola cars to the oxide crusher yard. The locomotives uncouple, are sent to the empty car yard to make up a new train of gondola cars and then despatched to the ore loading shovels. The loaded gondola cars are pushed through the rotary car dumps in pairs or singly by special 30-t electric locomotives and run by gravity from the dumpers to the empty car yard.

The side-dump cars are made up into more or less permanent trains with a locomotive, are despatched to sulphide or waste loading shovels and then to the waste dumps or to the concentrator crushing plant as required.

A chief despatcher is in charge of all traffic movement. Traffic control is maintained by six despatchers in charge of intermediate stations at strategic points in the mine. The despatchers control the electric switches in their respective areas, intermediate stations being linked by telephone. The chief despatcher is in radio communication, through a dual system, with all locomotives and locomotive cranes. This system allows for the control of all train and locomotive movement in the mine and for re-routing traffic when necessary.

In addition to the rail haulage system for ore and waste, a fleet of diesel-engined trucks is used for haulage of waste. There are 48 units in service, twelve of these being new 70-t units with 600-hp engines of the semi-trailer rear-dump type. Eight additional such units have recently been acquired. Of the remainder, seventeen are of the single-axle end-dump type of 25-t capacity, four are of the single-axle end-dump type of 15-t capacity and five are of the tandem-axle end-dump type of 30-t capacity.

Material hauled in the month of July 1964 by both rail and truck totalled 4 970 000 t in 27 working days and comprised of oxide ore 1 208 069 t; mixed ore 67 024 t; sulphide ore 1 506 711 t; and waste 2 188 196 t. The waste to ore ratio was 0·787 to 1 and the average material mined was 184 074 t/day.

Of this total, 3 349 050 t were moved by rail and 1 620 950 t by truck. Total ton-kilometres by rail was 19 450 376 and for the trucks 3 081 559.

Truck haulage is confined to haulage of waste from the upper benches of the mine, where the haulage distance is short. Dumps used for the disposal of waste by rail are arranged so that the side-dump railroad cars dump over the berm of the dump, the track being moved to the edge of the dump as required. A spreader plough hauled by a locomotive is used to keep the berm clean. Dumps used for waste disposal by truck have access roads at a maximum grade of 8 per cent. The trucks dump over the berm of the dumps, kept clean and smooth by bulldozers.

Fifty-three service trucks are in use for the delivery of water, fuel and personnel around the mine, as well as for explosives, stemming and general maintenance purposes. There are thirteen tractors equipped as bulldozers and four road-graders in service.

Maintenance of mine equipment is done in a number of shops that have been provided as requirements have arisen. Centralized shops are being planned and construction will be started during the coming year.

FUTURE MINING PLANS

The mining of the Chiquicamata orebody is in a phase of change. Both oxide and sulphide ore development and delivery to the plant must be balanced against plant capacity and production requirements. In the next five or six years the majority of the oxide ore reserve will be depleted and production of copper will largely depend upon sulphide ores alone. Most of the sulphide ore lies at elevations of 235 to 1100 ft below the present location of the primary crushing plants.

The retention of open pit-mining methods will depend largely upon the development of haulage equipment. Present planning is to continue with rail haulage of ore and, in order to do this, the grade of the entry tracks to the lower benches is to be changed from 3 to 5 per cent. This will require heavy, high-powered helper-locomotives to assist the present locomotives up these grades to the track system close to the crushers. Waste will continue to be hauled by rail from lower elevations and by truck from the heavy development that is required on the west side of the orebody.

Other possibilities include the use of electric wheel trucks now in development to replace the railroad system and the moving of the primary crushers into the pit with conveyor belts delivering ore to the secondary crushers in the plants.

Undoubtedly, at some future date some form of underground block-caving method of mining will have to be used to obtain the ore from levels lower than are now contemplated. Block-caving was studied in great detail in the 1940s, but the high development cost of putting it into effect, as against the great improvements in equipment that could be used for open-pit mining led to the continuation by open-pit methods. Present planning is to continue mining the orebody by open-pit methods.

CONCLUSION

Chuquicamata mine contains one of the world's greatest reserves of copper, and its production of metallic copper is the largest from any single mining operation. Under the present methods of mining the cost of production is comparatively low, and through direct and indirect taxation provides a very substantial proportion of the revenues of the Chilean Government.

Law No. 11,828 passed by the Chilean Government in 1955 radically changed the method of taxation applicable to the large mining companies operating in Chile. The tax is levied on the operating profit of the Company's operation on a sliding scale, depending upon the production of the mine over a basic one, taken as 95 per cent of the production for the years 1949–1953.

The new tax was to be 50 per cent of the operating profit plus a variable 25 per cent surtax, which drops to zero if the basic production rate is doubled. Under the same law the bank rate at which the Company could exchange U.S. currency for its operation was made more liberal. Taxation relief was given under the law to encourage the construction of refining and manufacturing facilities in Chile. Other conditions of this law required the formation of a Fiscal Copper Department whose function is to control purchases and import permits to ensure that the maximum advantage is taken of purchasing operating materials and supplies within Chile. Under the provisions of this law considerable investment has been made in the mine and plant and has resulted in an increase in copper production from the 1949–1953 average of 339 654 500 lb per year to the present figure of approximately 600 000 000 lb per year.

Since Law No. 11,828 was made, additional laws have been passed which have increased the direct taxation as originally stipulated.

Planning and investment in the mine, now actively under way, will raise the waste stripping ratio to 0·9 to 1·0. The main entry railroad ramps to the lower benches of the mine are to be changed to 5 per cent grade. One additional shovel has been acquired and several additional 70-t trucks.

These investments will ensure that the mine continues to provide the copper in the quantity necessary to maintain its position as the world's foremost copper producer.

Discussion

(Papers 31–34)

Mr. P. Best,* introducing Papers 31 to 34, said that 'Stripping operations of the Demerara Bauxite Co., Ltd., British Guiana', by Mr. W. Forbes, described the Company's operations in the production of metallurgical, refractory, abrasive, and chemical grades of bauxite for export to thirty different countries, centred at Mackenzie, some 65 miles from Georgetown, on the Demerara River.

In describing the port facilities, the author drew attention to the limiting draught of 19 ft which, of course, made loading difficult. That disability was overcome by 'topping up' bauxite ships at Trinidad and the alumina ships at Jamaica.

Although the deposits had been known since 1898, it needed the war conditions in 1914 to stimulate mining. Production in the early years was 3000 tons a year, which bore little relation to the present annual production of 3 500 000 tons.

A fascinating account was given of the improvements in stripping methods employed over the years for removing unconsolidated quartz sands, sandy clay and clay overlying the deposit.

During the early years most of the ore lightly covered with overburden was removed as a matter of expediency, but the rapidly increasing overburden–ore ratio led to the adoption of large equipment. An adverse consideration in that trend was the rainfall, something like 100 in. in one short season of mainly torrential downpours, which not only caused spoil piles to slump and bury drainage ditches, but made the clay increasingly difficult to handle. One compensating advantage of the climatic conditions was the ample space available for disposing of waste material—without which a thickness of 200 ft of overburden would make the operation uneconomic if it had been situated in a more populated or developed area.

The earliest mechanical stripping was done by small power shovels and draglines loading into dump cars. Then power-shovel loading into track-mounted wagons hauled by tractors was introduced, and in the late 1930s that method was superseded by tractor- or self-propelled scrapers together with hydraulic stripping, which continued to uncover much of the ore up to 1945. The quantity of overburden removal required in the 1950s would have made that method too expensive without some reduction in the high maintenance costs being achieved. Moreover, the use of self-propelled vehicles for the removal of 40 ft of tough, slippery clay was not exactly an attractive proposition when, to avoid interference with the mining operation, disposal outside the mining limits would have involved hauls of up to half a mile.

---

*Mining Consultant.

To overcome that difficulty hydraulicking, first introduced in 1938, was continued, combined with sand pumping, until 1952, when one dragline was introduced for overburden removal. A bucket-wheel excavator, with 1½ miles of conveyor system, was commissioned in 1962 to replace hydraulicking and sand pumping in one particular section of the mine when the latter method became unattractive owing to the distance to the sand-dump discharge area.

The operating costs given for all those methods greatly added to the value of the paper.

Dragline side-casting of overburden started in 1952 and at present four machines were in use. The long distances travelled by the draglines were impressive—the author referred to three walks of 30 miles performed by one dragline—although, perhaps understandably, this caused some failure in the main frame beams. The author pointed out that the dragline was limited in its effective reach by the provision that had to be made for the railway track and drainage channel. He suggested that truck transport might possibly be the answer, in spite of the disadvantage that bauxite became progressively stickier each time it was handled.

The paper concluded with a forecast of the future stripping procedures. Those would probably lie in reducing, with bucket-wheel excavators, the thickness of the overburden to be handled by draglines.

In some places it was possible that stripping might be more economic by a wheel excavator working in conjunction with a dredge pump.

In 'Ranstad—a new Swedish opencast mine' Mr. G. Olsson described an opencast mine now in its pre-production stage and scheduled to come into production in 1965 to provide Sweden's domestic supply of U_3O_8 at the rate of 140 metric tons per annum. The ore was a low-grade shale—very low by accepted standards, about ½ lb per ton. The capital cost was estimated to be about £10 000 000.

The author mentioned several considerations of technical interest that have influenced the planning, probably the most important being the comparatively shallow overburden on the SE boundary which enabled the more economic method of opencasting to be employed for the first fifteen years of operation. It was then proposed to change to underground mining and in readiness for that phase the primary crusher had been installed underground, to which access would be afforded during the opencast period by means of two 17-ft by 18-ft haulage tunnels.

Cost maps, based on drilling and seismic information, had been prepared to show stripping costs for the whole of the opencast area; and with an average overburden:ore ratio of 3·5:1 it would be appreciated that that method of evaluation was particularly important.

Planning of that kind gave some indication of the comprehensive and detailed study involved in the operation.

The mining method by walking draglines was described in detail and particularly noteworthy were the considerations that went into the choice of machine suitable for the size of the job. There was the complication that the gaps between the sizes manufactured were very wide and the next larger-size machine to the one chosen would be nearly double the weight and price. Drilling would be by electro-hydraulic machines at present employed in the development work. When production got under way the third drill of that type, found to be very successful, would be employed. The uranium shale drill would have sufficient reserve capacity to supplement, where necessary, the overburden drilling. Trucks were to be of 20- to 30-ton capacity and the preparatory work, begun in 1960, should give the operators ample time to evaluate the performance of all the equipment that was to be used in production.

As was usual with Swedish operations, the utmost economy was being made in the personnel employed, and it was interesting to see that identical crews of two men would be trained to operate the dragline, changing over on alternate weeks to drilling and blasting and then going back to the dragline operation.

He very much hoped that the paper was only a forerunner of a further contribution which would give the actual costs since those quoted in the paper were tantalizingly incomplete. It was, nevertheless, a very valuable record of the detailed planning that had gone into a proposed operation.

'Open-pit mining at Chuquicamata' by Mr. R. C. Becker and Mr. E. W. Witcomb was a description of one of the really enormous opencast copper operations in Chile. He found it difficult to appreciate fully the scale of the tremendous operation, although Fig. 2 (Plate I) did convey some idea of the enormous area of the pit, 2 miles long and from 660 to 1400 yd wide and just over 1000 ft deep from the topmost of 24 benches.

The authors gave a very interesting account of the growth of the project and it was indeed a far cry from the original planned rate of 10 000 t/day and 100 000 000 lb of copper per year when work started in 1914 to the current annual production of 600 000 000 lb of copper, entailing the daily removal of 175 000 tons of ore and waste from the mine.

Present mining practice was described in detail. Initially, new benches were 43 ft high, within the range of the mechanical shovels discharging loaded material to rail-cars on the bench above. Once that bench was sufficiently wide to allow a track to be laid, shovels with shorter booms replaced those of longer range and with the similar development of a second 43-ft bench the first was eliminated, giving an eventual bench height of 86 ft. Many examples of improved practice were quoted, notably the replacement of 22 churn drills used for blast-hole drilling up to 1956 by 7 rotary drills giving the same footage.

Another significant advance in technique was the introduction of AN–FO as the standard explosive, which reduced by almost half the amount of secondary blasting. The blasts were of a size commensurate with the vast operation and as much as 1 700 000 tons had been broken in one blast, although the average was 200 000 to 250 000 tons, for which 46 tons of explosive in 60 to 85 blast-holes were needed. Twenty-four mechanical shovels, all electrically powered, were in use, de-rated because of the difficult digging conditions. The trend in recent purchases had been towards larger buckets of 8- and 11-yd^3 capacity.

Servicing, of paramount importance in any large operation, was described very fully. Fifty-seven service trucks looked after ancillary needs and central workshops were being planned to take the place of many maintenance stations scattered around the property.

The authors closed with brief comments on future mining plans. The main factor there was the depletion of the oxide reserves in the next five or six years. Future production would come entirely from sulphide ore, the reserves of which lay at elevations of 235 to 1100 ft below the present primary crushing plants. The intention was to continue with open-pit methods with rail haulage, and increase the entry gradients to the lower benches from 3 to 5 per cent, which would call for heavy and powerful locomotives to assist the present locomotives to the crushers. It was possible that the rail system would be replaced eventually by the electric-wheel trucks now being developed and also that the primary crushers would be moved into the pit with conveyor belts delivering ore to the secondary crushers in the plants. Block-caving had been studied, but its introduction had been postponed owing to the high development costs involved.

Particularly interesting was the operation of the present Chilean taxation legislation referred to in the last paragraphs of the paper.

'Selective mining practice at Mary Kathleen uranium mine, Queensland' by Mr. D. L. Munro was of more than passing interest to their present Chairman and to himself since they were both connected fairly closely with the project in its early days. The deposit itself was a very complex geological feature. Garnetization was an important phenomenon in that all the uranium mineralization was associated with selective replacement of the intensely garnetized zones by rare-earth and uranium minerals, and that appeared to be controlled by a system of the pre-existing joints.

The initial proving of the deposit was done by diamond drilling with vertical and inclined holes 300 to 500 ft deep on a grid, together with costeaning to examine surface exposures and an exploratory adit. One advantage of the radioactive nature of the orebody was to facilitate the rapid and positive evaluation of the deposit by geiger instruments. The nature of the internal structure was determined by monitoring the diamond-drill core with a hand-held geiger counter at 6-in intervals, and that was checked by probing the holes and taking readings at 1-ft intervals with a small geiger counter protected by a steel sheath. The assaying was almost entirely done by radiometric methods, the equipment being calibrated against chemical standards, with chemical checks on selected samples.

The irregular nature of the ore dictated not only the mining practice—underground mining was impractical because of the selective mining needed—but also the size of the equipment used in the opencast. Large equipment and low-cost opencast methods were thus ruled out and the operation was accordingly planned on a standard 25-ft bench and with $1\frac{1}{2}$-yd^3 shovels.

The author described the selective mining method in detail and although on first reading it gave the impression of being time-consuming and impracticable, that was very far from the case and, once a routine was established, it functioned reasonably effectively in practice.

All vertical blast-holes for primary breaking had to serve the dual function of breaking ground and providing a sample of the ground intersected. Those holes were therefore surveyed and probed at 1-ft intervals as a matter of daily routine to decide which portions were ore or waste. The logs provided information also of the ore attitude behind the face and were used in conjunction with most meticulous face mapping to direct selective firings.

After the marking and sorting the blast, the obvious waste, something like 150 to 200 tons per day, was removed by 5-ton mobile crane.

Final evaluation was obtained by passing the Euclid truck loads through the combined discriminator and weighbridge as they left the opencast. The discriminator consisted of a series of six geiger scanning tubes arranged above the load in the form of a parabola, which gave an average reading of the uranium content of the load. The operator of the equipment could direct loads either to waste dumps or to one of the stockpiles according to grade.

In evaluating the results the author referred to the difficulty in determining the effectiveness of selective mining where there were no absolute standards against which mining performance could be measured.

From the author's personal observations the worst 6-month period of low grade and maximum dilution coincided with the time when selective mining procedures were relaxed. Towards the end of the production period it became possible to make estimates that were very close to the actual production figure by applying correction factors of +33 per cent to the probe tonnage and −19 per cent to the probe content.

The paper described the development of a successful selective mining system for an unusually irregular mineral occurrence. It would be interesting to see how the procedure would be adapted and modified when the world uranium demand enabled the Mary Kathleen mine, now 'mothballed', to resume operations.

Mr. P. Westerberg,* referring to Mr. Olsson's paper, asked for details of the sampling methods used to prove the deposit and for information regarding the sampling procedure adopted to control day-to-day production operations.

If drill-hole sampling were used, perhaps details could be given of the spacing of drill-holes and the number of samples taken per ton of ore *developed*, and the cost, and similar figures, per ton of ore *mined.*

Mr. J. Taylor† asked Mr. Olsson if any geological control, by radiation monitoring or other method, were applied during the stripping and bonding operations. That could obviate dilution of the ore and its removal along with the waste shale.

Leaching of uranium by acidic groundwater was a well established natural process. The limestone and shale overburden both contained pyrite so that the groundwater might well become acidified and promote leaching of the uranium. Was there any evidence that that took place and was the flow of groundwater likely to be great enough to carry a significant total amount of uranium at a level of, say, a few ppm?

Several other elements were said to be extracted along with the uranium during the acid leach process. He would like the author to identify those elements and indicate whether they had any value as by-products.

---

*Independent Mining Engineer.

†Geological Survey of Great Britain, Atomic Energy Division, London.

Mr. A. L. Austen* said that there were parallels to be drawn between Chuquicamata and Nchanga: among others, both mines produced a very high-grade and had very large reserves. Regarding ore reserves, perhaps the authors could clarify the position at Chuquicamata referred to on page 734. Did the figure of 595 000 000 tons at 1·59 per cent Cu apply to ore to be mined by opencast methods only, and was it independent of the figure of 375 000 000 tons at 1·73 per cent Cu.

Nchanga was probably the only mine with total mineral reserves approaching those of Chuquicamata, but the latter had already produced nearly 10 000 000 tons of copper.

Turning to the labour cost breakdown (Table I, p. 732), he thought that those not familiar with the operation would be surprised at the high rates of pay. Those, of course, were tied to the value of the escudo on the free market—originally at parity with the American dollar, but at far below par at present.

Interesting (and elsewhere rare) minerals were mined at Chuquicamata and following a visit to Chile he had been able to present to the Geological Society of Zambia the first specimen of antlerite to be seen there.

Mr. C. B. Campbell† said that Papers 31 and 33 concerned uranium mines of unusual interest. The Mary Kathleen deposit was probably the largest rare-earth skarn deposit in the world and, compared to the content of lanthanons, uranium was a minor constituent among the potential ore minerals in that orebody.

The deposit was found at outcrop by a syndicate of prospectors who realized that they had found something fairly 'big', but it was nearly a year before the 'backroom boys' could say what had in fact been found. Proving the deposit took two years from the time of discovery and, in the circumstances, that implied considerable thrust on the part of all concerned.

Apart from the difficulty of mining and treating such an unusual ore, the establishment of the enterprise in a remote part of Australia presented certain special difficulties. Those were alluded to very modestly in the paper. One difficulty not mentioned was the proximity of an already well established large base metal mine at Mt. Isa, 40 miles away from Mary Kathleen. Besides competing requirements for rail transport, there was the need to establish at the outset at Mary Kathleen all the amenities which had taken from 25 to 60 years to mature in other large Australian mining townships.

The two principal factors which probably ensured the profitable working of the Mary Kathleen orebody were the application of strict geological control in selective mining and the use of the electronic sorter. The latter was briefly referred to in the paper, but it was that device which gave a bonus to the operators' profits. Its application to the peculiar type of ore was an ideal one and its invention, in Canada, came not too late for the mine to derive enormous benefits.

---

*Formerly, Mining Engineer, British South Africa Co., Zambia.

†Metals Branch, United Kingdom Atomic Energy Authority, London.

The use of electronic monitoring equipment by samplers on the bench faces and in logging bore-holes was a special feature of the deposit. Although that sort of instrumentation was at present peculiar to uranium mining, the introduction of similar portable equipment in the next decade in other metal mines might well come about, making possible on-the-spot assays for several other elements.

Turning to Paper 33, the account of a new uranium mine in Sweden was of particular interest because it described the working of a very large tonnage, very low-grade orebody, of what had hitherto been regarded as sub-marginal material. There the emphasis was on economical mining and overburden removal rather than on grade control.

The alum shale in the section now being worked had an average grade of only 0·03 per cent U_3O_8. That was only a shade more than the average grade in the Witwatersrand gold–uranium deposits but, unlike the Rand deposits, uranium was the main and only product, not a by-product. That was the first attack on the category of reserves which might be called 'second line material', which included certain shales, phosphates and even sea water.

Practically every country boasting a Geological Survey had looked for another 'Blind River' or 'Witwatersrand' field within its borders. In spite of intensive searches and the comparative ease with which radioactive mineral occurrences might be found at outcrop, no major discovery of uranium had been made since 1955. The reserves of uranium were small compared to those of coal and oil and they gave cause for concern. The advent of Swedish shale, having a potential of up to 1 000 000 tons of U_3O_8, might now pave the way for other similar operations which should help to bridge the gap. Much depended on how low costs could be kept in mining and treating such very low-grade material. There would undoubtedly be an increasing demand for uranium in the next decade and thereafter, but if the price went too high it would price itself out of the market.

Mr. S. D. Day,* referring to Paper 31, said that one was left to read between the lines to visualize the many difficult problems which must have faced the engineers during the early planning stages and the subsequent carrying out of what had proved to be a most successful project.

---

*Fergusson, Wild and Co., Ltd., London.

It was therefore particularly unfortunate that the temporary lack of demand for uranium oxide had made it necessary for the mine to be placed on a care and maintenance basis after only five years of normal operation. Some of the Blind River uranium mines in Canada had been faced with the same problem.

Had any special steps been taken at Mary Kathleen to protect the fixed installations during the shutdown period? The Navy were reported to have developed a simple technique whereby equipment was sealed in a vacuum to prevent condensation and corrosion, and it would be interesting to learn if that type of protection could be applied when valuable mining and milling installations had to be closed down for a period of years.

Mr. T. T. Heywood,* said that Mr. Munro explained selective picking after a blast by the use of a crane. He wondered if a drop-ball had ever been used as an alternative to secondary blasting and, if so, with what results.

Mr. B. G. Fish† said that he wished to comment on two aspects of ammonium nitrate blasting, a subject referred to extensively during the Symposium both in the papers and in discussion. The first point concerned the problem of mixing. In his introduction of the papers in Session 3 (pp. 279–81) attention was drawn to the conditions permitted for ammonium nitrate application in the United Kingdom. There were two alternatives: either a licensed factory for central mixing, which was feasible only on the very largest operations, or local mixing at the bore-hole for immediate use. In the latter case a simple mixing trough was often employed, the ammonium nitrate being placed in the trough, the fuel oil poured on and the materials mixed with wooden paddles. The mixed AN–FO explosive was then tipped down the hole. An alternative permissible system was the use of a pneumatic mixer/loader, of which several varieties were commercially available.

In the literature many methods of mixing AN–FO were described—from very sophisticated mobile plants which intimately mixed the constituents as they were poured into the bore-hole to the less sophisticated method in which a sack of ammonium nitrate was opened and the fuel oil poured in. That was allowed to stand until migration of the fuel oil was reckoned to have been completed when the material was charged into the hole.

A novel technique of mixing had recently been applied in Norway. The ammonium nitrate was supplied in 50-kg bags with a double

---

*N. V. Mijn Maatschappij, Curaçao.
†Institute of Quarrying, London.

container, the inner one being of plastic. The fuel oil was injected into the bags by means of a five-pronged stainless steel fork having five hollow and perforated tines. Injection was rapid, the necessary migration of the fuel oil was soon achieved and the whole operation could be completed without moving the bags from the transport vehicle. That technique was due to K. Varn of Dalen Cement.

The second point to be considered arose from discussion at an earlier session when the question of whether the quarrying industry in the United Kingdom really wanted slurry explosive was posed. The answer was simple: at £50 a ton delivered in plastic bags the industry was decidedly interested; at £150 a ton it could work up little enthusiasm. In other words, if slurries were to be commercial they must be kept economic and at the same time offer significantly better characteristics than simple AN–FO. If, however, the full cost of factory processing, distribution and delivery were involved, it was difficult to see how they could be made preferentially attractive in price. In those circumstances it was surely better to make use of the full range of orthodox commercial explosive.

WRITTEN CONTRIBUTIONS

Mr. L. M. Dunstan:* With particular regard to Papers 31 and 33, I wonder if it is considered that long exposure to the radioactive orebodies in the mines constitutes a risk to workers' health. If so, how is the problem dealt with?

Professor Ir. P. Th. Velzeboer:† I would like to ask Mr. Olsson what effect, if any, drifting snow has on the dumping of overburden. Have wind directions likely to prevail during winter been taken into consideration during the planning stage of the mine?

The question of better fragmentation with AN–FO and slurry explosives has already been debated, but I would like to add that recent experiments clearly showed that shockwave energy has very great influence on the fragmentation. The same experiments have also indicated that air gaps between the explosive charge and the rock are detrimental to the energy of the shockwave in the rock.

One could therefore conclude that AN–FO explosives give a better fragmentation than normal explosives in rock blasting, but that slurry explosives give an even better fragmentation.

---

*Ministry of Housing and Local Government (Geology Section), London.

†Technical University, Delft, Holland.

Authors' Replies

(Papers 31–34)

Mr. D. L. Munro: In answer to Mr. S. D. Day's enquiry (p. 750), the area was first cleaned and instruments and ancillary equipment removed for storage elsewhere. Normal lubricants were replaced by special lubricants and surfaces were covered with a protective coating and, where possible, covered with a black plastic. Constant inspection is carried out by the small 'care and maintenance' staff retained at the mine, particular attention being paid to the control of termite infestation, which is a very real problem in Northern Australia.

Vacuum sealing of large mining and milling installations is not, in my opinion, a practical proposition. In this connection, however, it is worth noting that the relatively arid climate—with low relative humidity and 17-in rainfall, occurring in a few isolated storms—is probably the most significant factor preventing rapid deterioration of equipment.

The use of a drop-ball, queried by Mr. T. T. Heywood (p. 750), was considered, but was rejected on the grounds that the garnet–allanite composition of the ore and waste rock was too tough for such a method to be successful. A striking example of the toughness of the rock was provided by the waste dump in which large three- and five-ton blocks rolled and bounced without chipping during a fall of 100 ft. Further considerations involved the restricted space available in the open-cut and the need to maintain full access to all benches in the interests of selective mining.

Mr. L. M. Dunstan wonders (p. 751) if long exposure to the radioactive orebodies constitutes a risk to workers' health. Tests carried out during the life of the mine on selected personnel from areas considered most likely to be affected showed no adverse effects. The effect in an orebody worked by open-cut methods, exposed to the benefits of natural wind circulation, would be less than in underground workings.

Mr. G. Olsson: In reply to the question by Mr. P. Westerberg (p. 747), the area suitable for open-pit mining and the adjacent underground mining area have been surveyed by diamond core drilling. The cores have been investigated by radiation test devices for evaluating the U content across the scale horizon. Calibration tests by chemical analyses have been performed.

The drilling pattern was 200 × 200 m in the most interesting parts; for the more regional surveying no regular pattern was used.

For checking the variations of U content during production we are experimenting with a logging device for use in the blast-holes.

In reply to Mr. J. Taylor (p. 747), during the stripping in the box-cut we have used the geological conditions to find the upper border of the rich zone by blasting down to a layer of limestone lenses separating the waste shale and the uranium shale.

Regarding the question of natural leaching of uranium shale by groundwater, we think this would be possible if blasted shale were to lie for a long time in the water flow. However, this reaction is not believed to have any practical importance.

The leaching process in the plant gives a solution containing about 0·6 g/l U. In addition, it contains:

	approx. g/l		approx. g/l
Fe^{2+}	9·3	V	0·18
Fe^{3+}	0·6	As	0·12
Al	0·5	K	4·3
Mg	2·9	Rare earths	0·28
Mn	0·3	PO_4 and others	5·3
Ni	0·2		

It is possible that some of these elements could be used in the future as by-products.

Professor Velzeboer enquires if drifting snow has any effect on the dumping of overburden (p. 751). Work with draglines is indeed curtailed by drifting snow, as well as by thick fog. Frozen overburden in the cut and frozen lumps of soil accumulating under the bottom plate are also problems encountered during the winter; they too contribute to the general diminution in efficient working during the cold season.

REMARKS AT CLOSING SESSION

Mr. D. S. Burwood: Mr. R. F. Lethbridge has kindly undertaken the onerous task of drawing together the highlights of the proceedings and I call upon him to present his review.

Mr. R. F. Lethbridge: Bringing together, as it does, members of the three Institutions dealing with 'open air' mining, this symposium has been truly oecumenical in its nature. I am sure we all realize now the great similarities that exist between our various 'faiths', rather than the differences. If, therefore, I refer to 'opencast' rather too often, please take it as referring to all our disciplines.

Thirty-four papers have been presented; and they cover aspects of opencast operations in the United Kingdom and other countries —ranging from Australia to the Arctic.

About 400 registrations have been made, of which one hundred or so are from abroad, 26 countries being represented.

Each one of you will take away his own memories of these crowded days but, if I may, I should like to mention a few of the, to me, outstanding impressions of the symposium. First of all, while not strictly a part of the symposium, the Sixth Sir Julius Wernher Memorial Lecture by Mr. R. P. Koenig gave us the broad canvas of the mining industry against which to measure the detail of our particular operations.* It was indeed a memorable evening.

Turning to the group of papers covering United Kingdom activities, I am sure we were all impressed by the sheer size and range of the work being done. The ingenuity shown in finding methods to work within the limits imposed by the various authorities gives much food for thought. Much interest was shown in the use of AN–FO and slurries and in ground stabilization, as well as land restoration.

Many of the papers are of a descriptive nature and will form, with the contributions from the floor, an invaluable record. I am sure you will all agree that these, together with their discussion, will form a much needed reference book and one that will be widely used for years to come. Such operations as, for example, the world's largest copper mine, make fascinating reading.

In papers and discussion dealing with the monster machines now in use in the U.S.A. and Germany, one was brought to realize how the interdependence of loader and carrier led from one increase in size to

---

*Koenig, R. P. Vertical integration in mining industry. *Bull. Instn Min. Metall.* no. 697, Dec. 1964, 106–26.

another, with emphasis on the need for maximum usage tending to favour continuous working where conditions permit.

The subject of the restoration of land worked out by surface methods was obviously in the forefront of our thoughts. It is clear how important this has become in opencast working, more particularly in countries where good land is scarce.

Both in the papers and in the discussion the essential role of scientific planning was very clearly brought out. In my Army days we were taught that 'time spent in reconnaissance is never wasted'. Preliminary investigation by all the techniques at our command and the detailed planning of all facets of an operation is now a *sine qua non* of successful open-pit operation and when all the horses arrive at the winning post together all concerned have real cause for satisfaction.

The paper on submarine operations caused much interest, if only in drawing our attention to the problems that lie not in the realms of fancy but in the immediately foreseeable future. If, as Paley says, our land reserves are running out, it is to the sea that we must turn in hope.

I sensed throughout the papers and discussion an eagerness to know about and to adopt new techniques. The widespread use of AN–FO and slurry for breaking rock; experimentation in angle holes; improvements in drilling techniques; aerial survey; photogrammetry; rock and soil mechanics; computer usage and work study—all were referred to and discussed in striving after higher efficiency and better costs.

In contrast to new techniques, it seemed to me, perhaps as too great a generalization, that we were still using old tools. The percussion drill, the shovel and dragline and wheeled transport still dominate the scene. Improvements in size, efficiency and techniques—yes; but still the same basic tools that we have used for a century. Someone said '90 years of percussion drilling and no advance!'

Signs of an awareness of this were there: jet piercing of rock (although this may not be the right line of approach); suction dredging; mobile crushing plant; belt transport; etc. One or two new ideas but nothing really fundamental yet. If we are to exploit the riches of the Continental Shelf and go forward into a new element we must develop not only improved but revolutionary tools as well as techniques. This is a challenge to us all.

May I add a personal note here? I had the interest and excitement of being connected with two of the properties which gave rise to papers—Miferma and Mary Kathleen. One leaves a little of one's heart in such places and both papers brought me very happy memories.

I think it should be emphasized that the success—and I have no hesitation in saying so—the success of this symposium has not only been due to the excellence of the papers and of their presentation but also to the members whose stimulating questions and informative replies and descriptions of relevant operations in other parts of the world all combined to make the time spent here both enjoyable and rewarding.

No symposium is alive without its members. We have had the opportunity of renewing old friendships and making new ones, we have heard first-hand comment, question and answer on a wide variety of subjects, not only in the formal sessions but at the informal ones that have taken place between those interested in special aspects of our deliberations.

It only remains for me to apologize for any sins of omission or commission in this summary, to thank you for your patience, and to offer particular thanks to Lord Fleck, who kindly agreed to undertake the task of officially opening the symposium, and to all who have contributed by papers or discussion.

Mr. A. Choubersky: May I take it upon myself, in the name of all the authors and of the members of this symposium, first of all to thank Mr. Lethbridge for his very able summary of the highlights of the technical sessions.

I am sure I am speaking for everyone when I say this symposium has been pleasurable, enjoyable, timely and useful. This has been due, in the first place, to all the work put in by the Organizing Committee which, under the Chairmanship of Professor J. H. Taylor, and with the help of representatives of the Institution of Mining Engineers and the Institute of Quarrying, has taken over two years to co-ordinate all the papers and to hold the many meetings necessary for arranging the symposium. Furthermore, we thank all the Chairmen and Rapporteurs who have conducted the sessions and have been the 'band leaders'.

Finally, I should like also to thank, on behalf of all of us, the staff of the Institution of Mining and Metallurgy who had the responsibility of organizing the details of the symposium, of preparing the papers, reading the proofs and co-ordinating everything.

I propose a hearty vote of thanks to the Organizing Committee, the Chairmen, the Rapporteurs, and the Staff of the Institution of Mining and Metallurgy.

Professor J. H. Taylor: May I very briefly express, on behalf of the Organizing Committee, our thanks to Mr. Choubersky for the kind things he has just said. Organizing the symposium has really been a most pleasant task, and I think the happiest aspect of it has been the cordial and friendly co-operation between three institutions with a common interest—the Institution of Mining Engineers, the Institute of Quarrying and the Institution of Mining and Metallurgy. One hopes that this is only a beginning of collaboration which will increase throughout the years.

Our task has been made easier by the tremendous help and support we have had, as always, from the staff of the Institution of Mining and Metallurgy.

I think that perhaps one thing the Organizing Committee may congratulate itself upon has been its choice of authors—for it is upon them that the success of any symposium depends. So long as we remain successful in obtaining good authors and stimulating and critical contributions to the discussions we shall not be afraid to undertake the organization of similar meetings in the future.

Mr. D. S. Burwood: I would add my personal thanks to Professor Taylor and to the members of the Organizing Committee jointly representing the three institutions. As we have heard, the symposium has taken two years to prepare. A symposium such as this takes a considerable time to organize, and it is only by the accident of the calendar that I have had the privilege and honour of presiding. I regret very much that their duties have prevented the President of the Institution of Mining Engineers, Mr. G. C. Payne, and the President of the Institute of Quarrying, Mr. F. Frith, from participating with me in this closing session. The symposium has, I think, emphasized the many lines of technical interest which are common to the three Institutions.

Delegates will shortly disperse. May I wish you all happy and safe return journeys. Many of you will be participating in one or other of the tours to visit the activities in this country which are relevant to the subject of the symposium and you will see some of the operations which have been discussed. These, I am sure, will be found of great interest, and I take this opportunity of thanking those organizations which have undertaken to receive delegates and to arrange these tours, which are so valuable in connection with a meeting such as this.

Gentlemen, I will delay you no longer. I thank personally those who have attended and contributed to the proceedings and, in particular, those who have travelled long distances to do so.

I now formally declare closed the Symposium on Opencast Mining, Quarrying and Alluvial Mining.

Personal Index

Alder, H. L. 113
Allen, H. E. K. 438
Anderson, E. 710
Archer, A. A. 437
Atkinson, T. 505, 534, 545
Audibert, J. 449, 539
Austen, A. L. 748

Ball, A. E. 416, 430
Barclay, T. F. 354
Bartz, J. 38
Baster, L. R. 692
Batt, V. G. W. 148
Batzer, D. J. 97, 196, 216
Bauer, A. 239
Beaton, G. M. 210, 214, 223
Becker, R. C. 727, 744
Beckman, W. C. 147
Bell, W. B. 531, 658, 670, 678
Bennett, R. B. 531
Bergne, J. A'C. 79, 87, 204, 209, 213, 222
Berman, L. 531
Best, P. 742
Beulaygue, M. 358, 359
Bishop, A. W. 389, 432
Black, R. A. L. 435, 443
Boulter, G. 574
Bouma, A. 221
Bradford, E. F. 98, 113
Brealey, S. C. 355, 390, 415, 434, 435, 436, 437, 439, 442, 531
Brindle, N. 79, 86, 87
Broadhurst, J. K. 97, 196, 216
Brooke, M. A. 84, 90, 205, 215, 222
Brunner, J. 261
Buffrey, R. 666
Burrage, A. C. 727
Burwood, D. S. 755, 758

Cameron, N. R. 675, 678
Campbell, C. B. 748
Carr, T. H. 531
Caruel, P. 449, 539
Chaplin, E. S. 673, 677
Chaston, I. R. M. 199, 200, 215, 216, 217, 220
Cheshire, M. A. 228, 239
Chiswell, J. 534
Choubersky, A. 449, 537, 539, 757
Collin, A. 431
Collins, S. V. 143, 148
Cooke, S. C. 531
Cooper, H. H. 415
Cotton, H. 531
Coursin, A. 542
Cruickshank, M. J. 125, 147, 198, 199, 203, 204, 205, 221

Dales, H. B. S. 285, 296, 437, 443
Daniells, J. H. 416, 430
Danjoy, J. J. 603, 610
D'Arcy 431
Davey, J. C. 538, 539
Dawe, H. G. 493, 533, 534
Dawkins, B. 434
Day, S. D. 749, 752
Denyer, J. E. 196
Dobson, P. W. 77, 90, 211, 222
Dover, T. M. 282, 351, 355, 551, 601, 603, 605, 608, 609, 610
Dron, R. W. N. 286, 290, 294
Dubnie, A. 601
Dunn, J. A. 199
Dunstan, L. M. 751, 752
Du Vaure, C. B. 124

Eden, W. J. 389
Edmond, T. W. 415
Edwards, W. 263
Evans, W. D. 59, 73

Fern, K. A. 440, 442, 446, 537, 541
Fish, B. G. 279, 290, 536, 545, 750
Flachsenberg, P. 299, 351, 353, 354, 356, 357, 358, 359
Forbes, W. 693, 742
Frank, M. 38
Frith, F. 758
Frush, C. O. 608

Gaffert, G. A. 531
Gatliff, P. W. R. 531
Geissel, R. L. 564
Gelin, R. 726
George, T. N. 263
Glossop, R. 389, 429, 437, 442, 443
Gorges, R. A. 608, 611, 613
Gray, P. G. J. 82, 93
Greenland, B. J. 673, 676

Hamilton, D. E. 531
Harris, P. R. 74
Harrison, H. L. H. 97, 98, 113
Harrison, J. 694
Harriss, K. W. 83, 94
Heinz, W. D. 531
Hemmant, D. G. 607, 609, 614, 667
Henkel, D. J. 369, 376, 382, 389, 431, 436, 438, 439, 441
Hettwer, A. 228, 239
Heyll, H. 492
Heywood, T. T. 282, 295, 354, 670, 678, 750, 752
Hill, J. C. C. 39, 56, 75, 85, 90, 208, 213, 214, 223
Hood, J. C. 356, 357, 360, 436, 441
Hormander, O. 726

Ingham, F. T. 98, 113

Jackson, R. G. 260, 279, 280, 281, 282, 283, 285, 288, 289, 295
Jacob, C. E. 400, 415
John, K. W. 376, 389
Jones, T. W. 584, 602, 605, 606, 614

Keeble, P. 148
Kenna, R. J. B. 355, 360
Kirkland, R. W. 363, 429
Kochanowsky, B. J. 227, 238, 279, 281, 282, 283, 284, 286, 287, 288, 290, 291
Koenig, R. P. 755
Konz, F. 20, 75, 77, 78, 89

Lancaster, J. G. 289, 296, 356, 359, 607, 611
Legget, R. F. 389
Lethbridge, R. F. 208, 286, 537, 606, 755, 757
Lim, L. S. 219
Little, A. L. 389
Low, D. W. 135, 148
Lukeš, J. 240, 280, 282, 285, 294
Lummel, C. J. A. van 209, 218

McDonald, Alison J. 219
McDonald, G. C. R. 219
McFarland, W. H. S. 180, 196, 197
Mackintosh, I. B. 239
McLean, I. H. 57, 78, 86, 92, 203, 221, 222
Mahadkar, S. 287, 292, 294, 296
Major-Stevenson, A. 357, 564, 609, 611, 612, 670, 678
Martin, W. G. 617, 659, 669, 671, 672, 673, 675, 676
Meigh, A. C. 369, 389, 431, 436, 438, 439, 441
Mel'nikov, N. N. 575, 602
Mel'nikov, N. V. 293, 294, 575, 602
Mero, J. L. 56, 147, 148, 224
Miles, E. T. 206
Miles, T. E. 206
Mohr, R. 214
Mond, L. 261
More, G. A. 40, 56
Morgenstern, N. 389, 432
Morley, L. C. 574
Morris, D. B. 81, 88
Morris, R. O. 289, 296, 439, 446, 675, 677
Mukai, M. 439, 441
Munro, D. L. 681, 745, 750, 752

Novozhilov, M. G. 344, 353, 354, 355

Oliver, V. H. R. 438, 441, 609
Olsson, G. 711, 743, 747, 751, 752
O'Neill, P. H. 156, 197, 201, 213, 219
Orr, K. S. W. 672, 676
O'Sullivan, P. E. 531, 658, 670, 678

Pakianathan, S. 114, 198
Paley, W. S. 199
Pardoe, S. W. L. 606, 614, 670, 676
Parker, C. B. 537, 541, 546
Paterson, D. S. 439
Paterson, T. D. 57, 92
Payne, G. C. 758
Peck, R. B. 415
Peele, R. 147
Pentz, D. L. 535, 543
Posnansky, M. 59
Potter, W. 80, 87, 88
Price, V. E. 389
Protodyakonov, M. M. 345, 346, 356
Pryor, R. N. 532, 545

Rawcliffe, E. 83, 88, 439, 442
Rawiel, R. F. 483, 534, 545
Rich, E. 436, 441, 605, 612
Richey, J. E. 389
Richter, G. 492
Riley, G. G. 283, 290, 295
Roberts, E. A. 531
Roessler, E. B. 113
Romanowitz, C. M. 147, 148, 178, 223
Rushton, M. W. 285, 295, 674, 677
Ryan, B. C. 692

Saunders, D. W. 147, 203, 213
Schellhorn, H. W. 564
Schulte, H. 531
Schwidewsky, K. 492
Scott, E. 237
Scott, N. S. 56
Shadmon, A. 282, 290, 294, 535, 542, 605, 611
Shalowitz, A. L. 148
Shanklin, J. K. 81, 86, 87, 92
Sheil, G. 324, 352, 355, 356, 359
Shepard, F. 221
Sheppard, J. S. 202, 221
Shrimpton, G. J. 82, 87, 538, 540, 675, 676
Simpson, C. J. 208, 210, 214, 220, 223
Simpson, P. 114, 198
Skipp, B. O. 285, 294
Smart, L. A. 206
Smith, B. 263
Snijders, P. A. 710
Speake, J. G. 287, 291
Speight, D. E. 670, 678
Stanton, R. E. 219
Stewart, A. L. 214
Stewart, E. D. J. 531
Stephenson, R. 433, 434
Stoetzel, D. 531
Stuart, W. T. 443
Stumpf, K. 323

Taylor, J. 747, 753
Taylor, J. H. 601, 605, 610, 614, 757, 758
Templeton, T. M. 284
Teply, E. 603, 610, 612, 613
Terzaghi, K. 415, 431
Theis, C. V. 415
Thielen, H. 531
Tilmann, W. 637, 667, 675, 677
Todd, D. K. 415
Trotter, F. M. 263

van Lummel, C. J. A.: *see* Lummel, C. J. A. van
Vann, G. 3, 74, 76, 78, 79, 80, 81, 82, 83, 86
Varn, K. 751
Velzeboer, P. Th. 751, 753
Verantius, F. 24
Verbeek, R. D. M. 210

Warburg, R. O. 80, 88
Ward, D. L. 281, 295, 537, 541

WARD, W. F. 158
WARD JONES, D. R. 607
WARING, R. H. 363, 429
WEIS, J. 574
WEISSKIRCHER, R. 605, 611, 613
WENZEL, L. K. 415
WESTERBERG, P. 747, 752
WHITE, J. W. 536, 542
WHITE, W. S. 389
WHITESIDE, R. 57, 77, 89, 92
WILLIAMS, F. A. 102, 113, 219
WILLIAMS, J. C. 140
WITCOMB, E. W. 727, 744
WONG, Y. L. 219
WOODHEAD, E. H. 531

ZEINDLER, R. W. 565, 601, 603, 605, 608, 612

Subject Index

Acorn Bank strip-mining coal site, 617 ff
Aerial survey, *see* Photogrammetry
'Air-decking', 288, 293–4
Air drilling survey, of sand and gravel deposits, 15–16
  cost of, 16
Airlifts, use of, in alluvial mining, 137–8, 213
Alluvial deposits, 3 ff
  diamond-bearing, mining of, 46 ff, 142–3
  gold-bearing, mining of, 156 ff, 180 ff, 197
  offshore, mining of, 39, 125 ff, 199, 202 ff, 209 ff, 221–4
  platinum-bearing, mining of, 156 ff, 197
  sand and gravel, 3 ff, 20 ff, 39 ff, 62 ff, 74 ff, 145–6
  tin-bearing, 97 ff, 114 ff, 196–7, 198, 199–201
Amang, 104, 105, 217–9
Ammonium nitrate, 265, 266, 284
AN–FO explosives, 236–8, 283, 284, 285, 286, 541, 622, 627, 673, 677, 685, 735, 750–1
Angle-drilling, for blasting, 228–31, 284, 287, 291, 292–3
Auger drilling survey, of sand and gravel deposits, 13–15
  by continuous-flight power auger, 14
  cost of, 13, 14
  by hand auger, 13
  by single-flight power auger, 13, 79
Avon Gorge, 379–82

Back-acter, use of, in evaluating sand and gravel deposits, 81–2, 87, 92
Bauxite, opencast mining of, stripping operations in, 693 ff
Beeston Terrace, 58, 59
Bench gravels, in Klondike District, Canada, 181
Bench height, in blasting, 231
Berbice Formation, 694
Blasting, 227 ff, 240 ff, 266–8, 328–9, 345–7, 685, 724–6
  (*see also* Corrective blasting, Deep-hole blasting, Heading blasting, Square blasting)
  with AN–FO explosives, 236–8, 283
  bench height, 231
  bore-hole diameter, 231 ff
  bore-hole pattern, 228–30
  burden, 231 ff
  costs of, 237
  drills for, 230–1, 286, 290
  with slurry explosives, 236–8, 283
  spacing in, 231 ff
Boglochty Series, 623
Bolting, to increase stability of rock slopes, 377, 436, 441
Bore-hole diameter, in blasting, 231 ff, 264–5, 290
Bore-hole pattern, in blasting, 228–30
Bottom-dumping barges, used with floating grab dredges, 32
Box corer, 221
Brown coal, opencast mining of, 637 ff, 667–9, 677–8
  costs in, 656–7
  dewatering in, 639–40, 654
  machinery used in, 640 ff
    as means of conveyance, 650–2
    for overburden disposal, 646–50
    for winning, 641–6
  restoration in, 655–6
  surveying in, 654
Bucket capacity, in cut-and-fill stripping of rock overburden, 561
Bucket and truck capacity, in quarry loading and haulage, 305–10, 322–3

Bucket-wheel excavator stripping, 702 ff
Bulldozers, 336–7
Bulldozing, in Klondike District, Canada, 192, 194
Bunter deposits, 3, 59, 60, 80
  assessment of, 78–9, 81, 86, 92
Bunter Pebble Beds, *see* Bunter deposits
Burden, in blasting, 231 ff

Caland suction dredge, 41
California dredges, use of, in mining of offshore alluvials, 132
Cansado new town, Kedia d'Idjil project, 470
Cassiterite, free, in alluvial tin deposits in Malaya, 102–4, 105, 106, 112, 201, 217–9
Caterpillar tractor and scraper, as topsoil remover, 62
C.E.G.B. fly ash, as restoration material in Trent Valley quarries, 70–1, 93
Cement sands, offshore, 144–5
Chamber blasting, *see* Heading blasting
Choco, Department of, Colombia, gold and platinum dredging in, 156 ff
  costs of, 168 ff, 201, 202, 220
  titles and concessions, 166–8
Chuquicamata, Chile, copper orebody, 733–4
Clamshell mining, of offshore alluvial deposits, 133–5
'Clarence B. Randall' cutter suction dredge, 42
Coal, opencast mining of, in Great Britain, 617 ff, 669–70, 671–2, 673–4, 676–7
  drainage in, 630–2
  economics of, 634–6
  excavation in, 618, 626, 627–8
  machinery used in, 618–21, 622, 626, 627–30
  output in, 634–5
  power supply in, 634
Coal (*contd.*)—
  preparation plant in, 632
  restoration in, 623, 630
Coal excavation, 622
Coal haulage, 622
Coal hoisting, 619–21
Coal preparation, 632
Coal screening, 622
Colliery refuse, as restoration material in Trent Valley quarries, 70
Computer, use of, in ore grading, 365 ff
Condoto River area, 166
Continental Shelf Act, 146, 202
Copper, opencast mining of, at Chuquicamata, Chile, 727 ff
  blasting in, 735–7
  drilling in, 735
  haulage in, 737–40
  labour costs in, 732, 748
  power plant in, 733
  water supply in, 732
Core measurement, in alluvial tin deposits in Malaya, 98–102, 205, 216–7
Corella Formation, Queensland, 682
Corrective blasting, 255–9
  computation of charges for, 257–9
Creek gravels, in Klondike District, Canada, 181
Crusher, at Kedia d'Idjil project, 464–5
Crushing plant, effect of, on quarry loading and haulage, 301–3, 354
Crushing plant, in limestone quarrying, 271–4
Cumin, definition of, 122
Curtain blasting, *see* Deep-hole blasting

Darcy's Law, 398
Deep-hole blasting, 240 ff, 295
  computation of charges for, 244 ff
Dewatering, in opencast brown coal mining, 639–40, 654
'Diamon', 266
Diamondiferous gravels, dredging of, 46 ff

Dinosaur dredge, 134–5, 203–4, 212
'Down-the-hole' drills, 462
Dragline excavators, in sand and gravel excavation, 63–5
Dragline mining, of offshore alluvial deposits, 133
Draglines, stripping, 557–60, 572, 700–2
Draglines, walking, use of, in opencast mining, 575 ff, 722–4
  in digging development cuts for drainage of deposits, 576
  in driving haulage cuts, 576–7
  in forming spoil banks, 577–8
  in loading rock into dump cars, 580–1
  in mining bedded deposits, 581–3
  in overburden removal, 580, 595
  in stockpiling, 578–80
Dredge installation, in South America, costs of, 168–9, 201–2, 220
Dredges, 213–5
  bucket-ladder, 213–5
  costs of, 168–9, 201–2, 220
  floating grab, for sand and gravel extraction, 23 ff
    anchoring and mooring of, 36
    costs of, 37
    design of, 24–31
    maintenance of, 37
    operating details of, 25
    processing plant in, 32
    transport systems associated with, 31–6
    winch unit system for, 28, 31
  hydraulic, 135–7, 213–5
  mineral suction, 39 ff, 84–5, 90–2
    Caland, 41
    design of, 40, 48 ff
    jet-venturi, 42 ff, 46 ff
    main specification of, 45
    treatment plant for, 48–51
    in underwater prospecting, 54–5
Dredging, 42 ff, 190, 193
  gold and platinum, in Colombia and Bolivia, 156 ff
    costs of, 174–6, 201–2, 220
    operating data on, 171–3
    recovery systems used in, 178
Drilling, of limestone, 264–6
Drilling, in opencast mining, 92, 159, 182, 328, 345, 724–6
  costs of, 161, 162, 163–5
Drilling survey, of sand and gravel deposits, 11 ff
  presentation of results of, 17–9
Drills, for blasting, 230–1, 286, 290
    costs of, 232
    'down-the-hole', 462
Drop coring, 221
Dulang, 102, 103
Dulang washing, 122
Dump radius, in cut-and-fill stripping of rock overburden, 561–2
Dynamic positioning, 129, 203, 221–2

Eastern Glaciation, 58, 59
Electrical equipment, in opencast mining, 524–5
Electrical power, in opencast mining, 505 ff
  distribution equipment, 521–3
  pit distribution of, 515 ff
    cable junction boxes in, 519
  primary distribution of, 511–5
Electrical resistivity method, of geophysical survey of sand and gravel deposits, 8–9, 78, 79, 81, 86, 87, 89
  cost of, 9, 88
'Engineering geology', 430–1 (*see also* Soil mechanics, applied to surface mining)
E.P.I., 130
'Erftsprung', the, 639
Evaluation of prospecting results, in Colombia and Bolivia, 165–6
Excavating machines, for stripping rock overburden, 555–60
  design of, 564
  draglines, 557–60
  shovels, 555–7
'Excavator quarry' method, of opencast mining, 582–3
Explosives, in opencast coal mining, 622, 627 (*see also* AN–FO explosives, Slurry explosives)

Face shovel, 314–7
F'Derik orebody, 452
Ferruginous quartzites, in Krivoi Rog, Ukraine, opencast mining of, 344 ff
  research in, 349–50
  technology of, 345–9
    blasting, 345–7
    drilling, 345
    excavation, 347
    piling of overburden, 349
    stripping, 345
    transport methods, 349
Floating conveyor systems, used with floating grab dredges, 32
Flood Plain deposits, 58–61
Footwall, the, Mount Morgan, Queensland, 327

Gas-exploder, 127, 205
Geological bench mapping in opencast uranium mining, 686–7
Geophysical survey, of sand and gravel deposits, 4 ff, 86, 87, 88
  methods of, 7–11
  presentation of results of, 17–9
Germany, opencast mining of brown coal in, 637 ff, 667–9, 677–8
Gold, opencast mining of, at Mount Morgan, Queensland, 324 ff
Gold-bearing alluvial deposits, mining of, 156 ff, 180 ff
Grading, of ore, *see* Ore grading
Gravel, *see* Sand and gravel
Gravel-pump mining, hydraulic, in Malaysia, 117 ff
Gravel pumps, in sand and gravel excavation, 63, 83, 94
Groundwater, 371, 372, 373, 388, 390 ff, 433–6, 437, 439, 440, 442–6
  confined, below mine floor, 395–8
    'heave', 395
    'piping', 398
  formulae, 398–401
  investigation of, 401–6
    by exploratory drilling, 402
    interpretation of results of, 406
    by pumping tests, 402–5
Groundwater (*contd.*)—
  at Neyveli lignite project, 406–15
  in overburden, 393–5
    pre-drainage of, 394–5
    spoil bank drainage of, 393–4
Grouting, 82, 87
  use of, in slope stability problems, 440, 442

Haulage, in opencast mining, 324 ff, 622, 685–6, 737–40
Haulage, in quarries, 269–71, 299 ff, 351 ff
  as affected by loading, 305 ff
  costs of, 303, 312–4, 359
  effect of crusher on, 301–3, 354
  effect of foreign inclusions on, 300
  effect of winning on, 303–4
  equipment for, 317–21
    number of trucks required, 310–2
    tipping gear in, 319
Hauser's formula, for computing blasting charges, 230
Hazen's formula, 402, 434
Heading blasting, 240 ff, 294
  computation of charges for, 244 ff
  preparation of quarry face for, 241–4
'Heave', 395
'Hi-Fix' position-indicating equipment, 204, 213, 222
Hilton Terrace, 58, 59
Holme Pierrepont deposits, 58, 60
Hydraulicking, of overburden, 183–4, 186–8, 193, 697–700
Hydraulic mining, in Klondike District, Canada, 190–2, 194
Hydraulic positioner, 230

Iron sands, offshore, 143–4
Ironstone Areas Restoration Board, 586
Ironstone Areas Special Development Order, 587

Jacob Modification Formula, 399, 406
Jet-flame drilling, 345

'Joseph L. Block' cutter suction dredge, 42
Jurajda's formula, for computing blasting charges, 251 ff, 255

Kati, definition of, 97
Kedia d'Idjil, 449 ff, 540 ff
  development of orebodies in, 457–60, 539–40
  exploration of, for water, 459
  geology of, 452–7, 544
  mine project, Tazadit, 460 ff, 532
    construction of, 480
    conveyor terminal and crusher, 464–5
    equipment, 461–3
    grade control at, 465–6
    loading at, 464
    pit design, 460–1
    pit-to-railway conveyor, 463–4
    planning of, 476 ff
    service buildings and plant, 466–7
  railway and port project, 467–70, 532
    construction of, 479–82
    planning of, 470 ff
    railway, 467–8
    transfer of ore, 468–9
Kinta Valley, 98
  geology and topography of, 114–6
  hydraulic gravel-pump mining in, 117–20
  water supply in, 116–7
Klondike District, Canada, 180 ff
  assessment of claims in, 182–3
  gravels worked in, 181
  mining tenure in, 181
  operations in, 183 ff
    bulldozing, 192, 194
    dredging, 190, 193
    hydraulic mining, 190–2, 194
    stripping, 183–4, 186–8, 193
    Sulphur–Australia ditch, 184–6
    thawing, 188–90, 193
Krivoi Rog iron ore basin, Ukraine, opencast mining in, 344 ff
Kullenberg core sampler, 145

Land restoration, *see* Restoration
Lares' formula, for computing blasting charges, 251
Lighting, in opencast mining, 525–6, 658 ff, 670, 678
  cost of, 663–4
Lighting, of railway sidings, 665–6
Limestone, 260 ff
  blasting of, 266–8
  drilling of, 264–6
  geology of, 263–4
Limestone quarrying, in Great Britain, 260 ff
  crushing plant in, 271–4
  history of, 261–3
  loading in, 268–9
  organization in, 277–8
  safety in, 275–7
  scrubbing unit in, 273
  stockpiling in, 274–5
  transport in, 269–71, 295
Linda Fault, Mount Morgan, Queensland, 327
Loading, in opencast mining, 324 ff, 685–6
Loading, in quarries, 268–9, 299 ff
  as affected by haulage, 305 ff
  costs of, 303, 312–4
  effect of crusher on, 301–3, 354
  effect of foreign inclusions on, 300
  effect of winning on, 303–4
  equipment for, 314–7
    face shovel, 314–7
    tractor loaders, 317
Local authority refuse, as restoration material in Trent Valley quarries, 70, 93
Loran, 130

Mackereth Sampler, 205
'Magnetorque', 316
Mary Kathleen uranium deposit, 681 ff
  geology of, 682–3
  proving of, 683–4
Masaboin Hill, Sierra Leone, 385–8
M.I. (Miner's Inch), 184
M.I.D. (Miner's Inch Day), 184
Mineral Workings Act, 588, 607

Mohole drilling project, 129
Mount Morgan opencast mine, Queensland, 324 ff
  bulldozers at, 336–7, 360
  drilling and blasting at, 328–9
  lowering equipment at, 334
  power shovels at, 329 ff
    hire cost of, 333
    maintenance of, 335–6, 360
    output of, 332
    overhaul of, 336
  trucks at, 337–42
    costs of, 340
    performance of, 339–40
Mount Morgan, Queensland, geology of, 327–8

'N–IV' porous prilled ammonium nitrate, 285
Narino, Department of, Colombia, gold and platinum dredging in, 156 ff
  costs, 168 ff, 201–2, 220
  titles and concessions, 166–8
Nechi River area, Colombia, gold and platinum dredging in, 156 ff
  costs, 168 ff, 201–2, 220
  titles and concessions, 166–8
Neyveli lignite project, South India, as example of groundwater control, 406 ff

Ocean minerals, 125–7, 154–5
  cement sands, 144–5
  on the continental shelves, 126–7
  diamond-bearing gravels, 142–3
  iron sands, 143–4
  on the ocean floor, 125–6
  precious metals, 140–1
  sand and gravel, 145–6, 202–3, 221
  sea-floor nodules, 146–7, 204
  tin-bearing sands, 141–2
Ochil Fault, 623
Offshore alluvial deposits, exploration of, 127–9
Offshore alluvial deposits, mining of, 130–8, 140 ff, 150–3, 210–1, 212
  by airlift, 137–8
Offshore alluvial deposits (*contd.*)—
  by bucket-ladder dredges, 130–3, 149
  by dragline and clamshell, 133–5, 149
  by hydraulic dredges, 135–7, 149
  legislation concerning, 138–40
  position location in, 129–30, 203, 204
  waste disposal in, 138
Opencast mining, 227 ff, 324 ff, 344 ff, 449 ff, 617 ff, 637 ff, 681 ff, 693 ff, 711 ff, 727 ff
  of bauxite, 693 ff
  of brown coal, 637 ff, 667–9
  of coal, 617 ff, 669–70, 671–2, 673–4, 676–7
  of copper, 727 ff
  determination of volumes in, by aerial survey, 483 ff
  drilling in, 328, 345, 724–6
  electrical equipment in, 524–5, 733
  electrical planning in, 505 ff
  'excavator quarry' method of, 582
  of ferruginous quartzites, 344 ff
  of gold, 324 ff
  haulage in, 324 ff, 622, 685–6, 737–40
  of ironstone, 584 ff
  at Kedia d'Idjil, 449 ff
  lighting in, 525–6, 658 ff, 670, 678
  loading in, 324 ff, 685–6
  power supplies in, 506–10
  stripping in, 551 ff, 565 ff, 618, 626, 627–8, 693 ff
  restoration in, 584 ff, 606, 608, 614, 623
  of uranium, 681 ff, 711 ff
  use of walking draglines in, 575 ff
  work study in, 416 ff
Ore grading, at Kedia d'Idjil, 459, 542
Ore grading, by computer, 365 ff
  cost of, 429
  programs for, 366–8
Ore grading, manual, 364
Overburden, 393–5 (*see also* Rock overburden)

Overburden (*contd.*)—
  crushing of, in opencast coal mining, 628, 672, 676
  disposal of, in opencast brown coal mining, 646–50
  spreading of, in opencast coal mining, 628–30
  stripping of, 41–2, 565 ff, 590 ff, 610–3, 626, 627, 693 ff, 716–8
    by dragline, 572, 580, 590–3, 611, 613
    economics of, 565 ff, 603–5, 610
    by hydraulicking, 183–4, 186–8, 193, 697–700
    by shovel, 573, 593–4, 611, 612
    by suction dredge, 41

Palong, 120–2
Pennine Drift, 58, 59
Percussion drilling survey, of sand and gravel deposits, 11–13
  cost of, 13, 78
Photogrammetry, 483 ff, 493 ff, 533–5
  costs of, 502–3, 545
  general survey of, 483–6
  in volume determination, 486 ff, 494 ff
    contours derived from, 487
    disadvantages of, 488–9
    in opencast mining, 497–9
    practical applications of, 489–92
    processing of data from, 489, 497
    in quarrying, 497–9
    random spot-heights derived from, 488, 499
    sections derived from, 487–8, 501
    of stockpiles, 499–500
'Piping', 398
Planning, 449 ff
  electrical, in opencast mining, 505 ff
  of Kedia d'Idjil project, 470–9
  of uranium opencast mine, Ranstad, Sweden, 720–2
Platinum-bearing alluvial deposits, mining of, 156 ff
'Point doctors', 189
'Point drivers', 189
Power plant, in opencast copper mining, at Chuquicamata, Chile, 733
Power shovels, 329 ff
Power supplies, in opencast mining, 506–10
Prospecting, for gold and platinum, in Colombia and Bolivia, 158 ff
  cost of, 160–5
Protodyakonov scale, 345, 346
Public Law 212, Outer Continental Shelf Lands Act, 139
Pumpwells, construction of, in groundwater investigation, 404

Quarrying, 227 ff, 279 ff, 299 ff
  blasting techniques in, 227 ff, 240 ff, 266–8
  haulage in, 269–71, 299 ff, 351 ff
  of limestone, 260 ff
  loading in, 268–9, 299 ff, 351 ff

Radford Factor, 100
Railway sidings, lighting of, 665–6
Range number equation, in ore grading, 367–8
Ranstad, Sweden, opencast uranium mine, 711 ff
  geological conditions at, 711–2
Reolite, 726
Restoration, 69–71, 78, 89, 93, 584 ff, 606, 608, 614, 623, 630, 655–6
Restored land, use of, 597–600
Rhine Valley rift, 20–3
Riss Glaciation, 23
Rock bolting, *see* Bolting
Rock mechanics, *see* Soil mechanics, applied to surface mining
Rock overburden, stripping of, 551 ff
  choice of machinery for, 551–3, 560–1
  cut-and-fill, 555, 560–2
  excavating machines for, 555–60
    draglines, 557–60
    shovels, 555–7
  instrumentation, 562–4

Rock overburden (*contd.*)—
  in opencast coal mining, 618
  transport of spoil from, 553–5
    by belt conveyors, 553–4
    by rail haulage, 553
    by truck haulage, 554–5
Rock slopes, 375 ff
  case studies of, 377 ff
  investigation of, 375–7
  stability of, 375, 391–3, 441–2
    improvement of, 377, 441
Rotary drilling, 82, 87

Sand and gravel, 3 ff, 74 ff
  dredging of, 20 ff, 39 ff
  extraction of, in Trent Valley, England, 62 ff
  location and evaluation of, 3 ff
  offshore, 145–6
Scaling factor, in ore grading, 367
Scrubbing unit, limestone crushing plant, 273, 295–6
Seismic methods, of geophysical survey of sand and gravel deposits, 9–10
  cost of, 10
Selective mining, of Mary Kathleen uranium deposit, 681 ff
Selective quarrying, 300
Shoran, 130
Shovels, 461
  stripping, 555–7, 573
Slewing cranes, used with floating grab dredges, 31, 32
Slide, the, Mount Morgan, Queensland, 327
Slope stability case studies, 377 ff, 439–40
Sluice-box, see *Palong*
Slurry explosives, 236–8, 283, 286, 290, 294, 541, 627, 685
'Soil-boring machines', 13
Soil mechanics, applied to surface mining, 369 ff (*see also* Rock slopes, Soils, properties of, Soil slopes)
Soil slopes, 371–5, 377 ff
  case studies of, 377 ff, 439–40
Soil slopes (*contd.*)—
  improvement of stability of, 377
  investigation of, 373–5
    by triaxial compression tests, 374–5
  stability of, 371 ff, 391–3, 441–2
Soils, properties of, 370–1, 437
Sonar survey, of offshore alluvials, 142
Spacing, in blasting, 231 ff
Sparker, *see* Gas-exploder
Spoil, from stripping of rock overburden, transport of, 553–5
Spoil banks, 393–4, 577–8, 609, 614
Square blasting, 241, 242, 243, 244
  computation of charges for, 244 ff
Stereo plotting instruments, 485, 486, 487, 489, 494 ff, 503, 533
Strip-mining, of coal, 617 ff
Stripping, of overburden, in opencast bauxite mining, British Guiana, 693 ff
  methods of, 697 ff
    bucket-wheel excavator stripping, 702 ff
      costs of, 710
    dragline stripping, 700–2
      costs of, 702
    hydraulicking, 697–8
      costs of, 698
    hydraulicking and sand pumping, 698–700
      costs of, 700
Sugar beet waste, as restoration material in Trent Valley quarries, 70, 71
Sulphur–Australia ditch, Klondike District, Canada, 184–6

Tailings, core measurement in, in alluvial tin deposits in Malaya, 100–2, 199–200
Tailings, disposal of, 123–4
Tailings, as source of loss of free cassiterite in alluvial tin deposits, 102, 103
Takong, 120, 122
Tazadit mine project, 460 ff

Tazadit orebody, 455
Terzaghi's definition of effective stress, 431
Thawing, in Klondike District, Canada, 188–90, 193
Theis formula, 398, 399
Thiem formula, 398, 406
Tin deposits, alluvial, 97 ff, 114 ff
  assay adjustment of, 104–6, 218
  bore grid in, 107–11
  core measurement in, 98–100
    in tailings, 100–2, 199–200, 216–7
  costs of prospecting, 112–3
  mining of, 114 ff
  valuation of, 97 ff
Tolerance factor, in ore grading, 367
Tractor loaders, 317
Transit, 130
Transport, in limestone quarries, 269–71, 295
Transport, of spoil from stripping of rock overburden, 553–5
Trent Valley, geology of, 58–61
Trent Valley gravel quarries, 57 ff
  costs, 68–9
  extraction in, method of, 63–6
  loading in, 68
  maintenance of, 72–3
  processing of raw material in, 66–8
  restoration in, 69–71, 78
  stocking in, 68
  topsoil and overburden removal in, 62, 93
  transport in, 71–2
Trial pits, in survey of sand and gravel deposits, 16
Triaxial compression tests, in soil slope analysis, 374–5, 382, 386

Underwater prospecting, by suction dredge, 54–5
Uranium, Mary Kathleen deposit, Queensland, opencast mining of, 681 ff
  assessment of truck loads in, 689–90
  calibration of grade assessment discriminator in, 690
Uranium (*contd.*)—
  drilling and blasting in, 685
  evaluation of results in, 690–2
  excavation design in, 684–5
  geological bench mapping in, 686–7
  loading and haulage in, 685–6
  muckpile marking and sorting in, 688–9
  primary selective firing in, 687–8
  secondary breaking in, 689
  survey and probing in, 686
Uranium, opencast mining of, at Ranstad, Sweden, 711 ff
  benching of uranium shale in, 718–9
  drainage in, 720
  drilling and blasting in, 724–6
    costs of, 726
  equipment in, 722
  haulage roads in, 719
  overburden removal in, 716–8
  plant process in, 712
  power supply in, 719–20
  selection of most suitable area in, 712–4
  use of dragline in, 722–4
    costs of, 724

Vallentine scale, 104, 105
Vauban's formula, for computing blasting charges, 250
Volume determination, by photogrammetry, 483 ff, 493 ff

Walking draglines, *see* Draglines
Ward drill, use of, in South America, 158–9
  costs of, 160 ff
Waste disposal, at Mount Morgan opencut mine, Queensland, 342
Westfield opencast coal site, 623 ff
'White Channel' gravels, in Klondike District, Canada, 181
Work study, applied to opencast mining, 416 ff
  to determine haulage operating costs, 420–2

Work study (*contd.*)—
  to determine haulage requirements, 417–20
  to determine productivity of railway transport, 422–8
Würm Glaciation, 23
Yukon Placer Mining Act, 181
Zouerate new town, Kedia d'Idjil project, 466

The Institution of Mining and Metallurgy

Founded in 1892 and incorporated by Royal Charter in 1915, for the advancement of the science and practice of Mining in respect of minerals other than coal, and of Metallurgy in respect of metals other than iron; and to afford the means for facilitating the acquisition and preservation of that knowledge which pertains to the profession of a Mining Engineer and Metallurgist

Full details of the Institution's publications and conditions for admission to membership can be obtained from the Secretary of the Institution, 44 Portland Place, London W.1.

Other I.M.M. Publications

Bulletin and Transactions

Mining Industry – Published in January, April, July, October. *Applied Earth Science* – Published in February, May, August, November. *Mineral Processing & Extractive Metallurgy* – Published in March, June, September, December.

Price £4 per Section; £10 for all three Sections

Transactions – An annual volume containing the 12 parts is also available.

Price £10

I.M.M. Abstracts

Published every two months, providing a world survey of current literature on economic geology, mining and metallurgy.

£2 per annum

Wire Ropes in Mines

Proceedings of a conference held in September 1950. 18 papers and discussion and summary of conclusions and recommendations.

Price £2 10s.

The Physical Chemistry of Copper Smelting by R. W. Ruddle.

A review of published information; published in 1953.

Price £1

Recent Developments in Mineral Dressing

Proceedings of a symposium held in London in September 1952. 39 papers and discussion.

Price £3

Mineral Resources Policy

Proceedings of a symposium held in London in September 1955. Six papers and discussion.

Price 10s 6d

Difficulties Encountered in Smelting in the Lead Blast Furnace by R. W. Ruddle. Published in 1957.

Price 7s 6d

Extraction and Refining of the Rarer Metals

Proceedings of a symposium held in London in March 1956. 22 papers and discussion.

Price £3

The Future of Non-Ferrous Mining in Great Britain and Ireland

Proceedings of a symposium held in London in September 1958. 24 papers and discussion.

Price £3

International Mineral Processing Congress 1960

Proceedings of a congress held in London in April 1960. 52 papers and discussion.

Price £5